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IRON MINING IN MINNESOTA

BY

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CHARLES E. VAN BARNEVELD

INTRODUCTION

This bulletin is published to meet the constant demands for both technical and popular information concerning mining practice on the Minnesota Iron Ranges. Field work was begun for this purpose in May, 1910. I wish to express my thanks to the officials of the various mining companies, to the local managers, superintendents, mine captains, and engineers, and to many others, for their unfailing courtesy and help.

Historical.—A few historical statements and figures are indispensable to a comprehension of the relative importance of the Minnesota Iron Ranges and of the magnitude of the Lake Superior district as a whole. The American production of iron for 1909 was 53,806,304 gross tons, 79 per cent of which came from the Lake Superior mines. The total American production of pig iron for the year was 25,795,000 (long) tons. Since Lake Superior ores (Western Missabe silicious ores excepted) are purer than the balance of the American product and only 1.9 tons of iron ore are required to make a ton of pig iron, this region furnished the raw material for over 82 per cent of the American pig iron production.

Actual production from the Lake region may be said to date from 1855, with 1,449 tons from the Marquette Range. During the 55 years ending Jan. 1, 1911, the production from the entire region reached the enormous figure of 493,000,000 tons. One-half of this is the product of the last 11 years, one-third of the last 7 years, and in 1910 the product was 8.77 per cent of the whole. During the 19 years of its life the Missabe Range produced 224,905,184 tons, roughly 45 per cent of the total Lake Superior production, and in 1910 the Missabe is credited with 29,201,700 tons, almost 70 per cent of the Lake Superior tonnage for the year, producing the raw material for approximately 40 per cent of the American and 17 per cent of the world's pig iron output.

It is interesting to note that the first trial shipment of Lake Superior iron ore was made from the Marquette Range in 1850, and in the same year the first discovery of iron ore was reported in Minnesota near Gunflint Lake. The first discovery of importance in Minnesota was made on the Vermilion Range in 1865. This resulted 17 years later in the organization of the Minnesota Iron Company to operate the Soudan Mine at Tower, and in the building of the Duluth and Iron Range Railroad over which the first Soudan shipment was made in 1884, nearly twenty years after discovery. The Ely district, credited with practically 22 million tons to Jan. 1, 1911,

was discovered in 1883 and commenced shipping from the Chandler Mine in August, 1888.

The history of iron mining in Minnesota may be said to have begun in 1881, when George C. Stone succeeded in interesting Charlemagne Tower, of Philadelphia, in the acquisition and development of the Soudan territory. These lands were on unsurveyed government territory and not subject to homestead unless they were surveyed. Special arrangements were made at Washington for the survey of T. 62, R. 15, known to be rich in mineral, and surrounding townships. Since any legal subdivision containing more than half its area in swamp land would become the property of the State and, therefore, not subject to entry, special care was taken so to manage the survey that this contingency did not arise. The next step was to select homesteaders who could be depended upon to convey their holdings at the proper time. Sioux scrip was used to a large extent. An indefeasible title to these lands was acquired in due time and in 1882 the Minnesota Iron Company, with a capitalization of ten million dollars, was formed, to take over these holdings. The development and financing of this property, situated in a wilderness, 75 miles from Lake Superior, called for faith, courage, and ability of the highest order. The promoters of the enterprise are said to have spent over three million dollars before success was attained.

The initial exploration and development of the Missabe Range is a monument to the courage, optimism, and industry of a few men who invaded and prospected under tremendous difficulties a most inhospitable wilderness composed of almost impenetrable forests, swamps, and barren, rock-strewn lands. Their efforts were rewarded in the early nineties by the discovery of several deposits of iron ore. These deposits upon development proved to be beds of very soft hematite of great area and relatively small depth, overlaid by varying thicknesses of glacial drift. So different were they from anything heretofore found in the Lake Superior region that for some time serious doubt was felt as to their economic value. At first the smelters objected to the ore on account of its texture, and it was considered unmarketable until that doubt was removed. When this question was settled, the explorers began to fear that the enormous tonnage revealed by their work would cheapen iron ore to the point of rendering their discoveries commercially valueless. Cheap mining methods involving large scale operations and cheap transportation to Lake ports then became problems pressing for immediate solution. The transportation problem was solved by building part of the Duluth and Missabe Railroad to connect with the present Great Northern system. This was accomplished by local men and largely by local capital.

The Missabe exploration boom began in 1890. For some time it was difficult to interest outside capital and there was much haphazard and wasteful mining. Capital finally began to come in, resulting in a marked improvement in transportation and a gradual, though much slower, improvement in mining methods. The advent of outside capital and the peculiar local conditions brought many industrial and financial problems to the front. The enormous ore-bodies were parcelled out and cut up into many small holdings. There was constant rivalry and contention among these interests. The individual was anxious to make the most of his holdings regardless of

the common interest. The individual need for quick returns on heavy investments led to selective mining, resulting in an enormous waste of resources. The lack of co-operation among individual producers on the one hand and between producer and consumer on the other brought inevitable demoralization of the market and, therefore, of the whole industry. When many of the small holdings were consolidated into a few strong concerns, the rivalry increased, the position of the remaining "Independents" or individual operators became worse, and for several years an industrial war was waged that brought loss and ruin to many.

In 1900 the Range had been pretty generally explored and it was apparent to many that resources reaching into hundreds of millions were in danger of being ruthlessly sacrificed on account of inability to conciliate the contending interests. Co-operation in the ordinary way being impossible, it seemed that the only hope of an economical administration of these vast resources lay in centralization of ownership and management. In 1901 the United States Steel Corporation was formed. A comparison of the present conditions on the Missabe Range with those existing during the thick of the fight would convince anyone of the benefits conferred upon a section of the country and upon an important industry by intelligent centralization of management and unlimited capital. Not only was there a cessation of internecine strife, but there was inaugurated for the entire Missabe Range a comprehensive and systematic plan of exploration and development based on the ultimate possibilities of the entire range.

Missabe ores are very variable in structure and composition. Ore-bodies, during exploration and development, are carefully graded and plans are worked out for the most advantageous marketing of the ore-body as a whole. Due weight is given to the fact that one ore-body may be the complement in grade, texture, etc., to one or perhaps several others. Without a judicious mixing of different ores from the same mine and often from several mines many ore-bodies would soon be depleted of the best ore and enormous reserves would become unmarketable for want of sufficient complemental ore to bring them to grade. In this connection it is interesting to note that the development of the enormous tonnage of this high grade, soft Missabe ore opened up a way to market some of the hitherto unsalable, low grade, silicious, low-phosphorous ores that occur so plentifully on the Marquette and Menominee Ranges. These ores supplied all the qualities which the Missabe ores lacked in order to make a desirable furnace mixture.

When the United States Steel Corporation entered the field, the range had been sufficiently explored and opened up to permit a fairly close approximation of the ore developed and a forecast of future possibilities. The corporation made a critical study of the situation and began the acquisition on an enormous scale of reserves for future generations. Its plans have always been made with a view to conservation of natural resources and the largest ultimate profit. Large areas were leased and enormous sums spent in exploration, in stripping ore-bodies preparatory to steam shovel mining, and in equipment. Mining practice was improved and gradually standardized. Transportation facilities were greatly improved and enlarged.

OUTLINE OF THE GEOLOGY OF THE IRON RANGES[1]

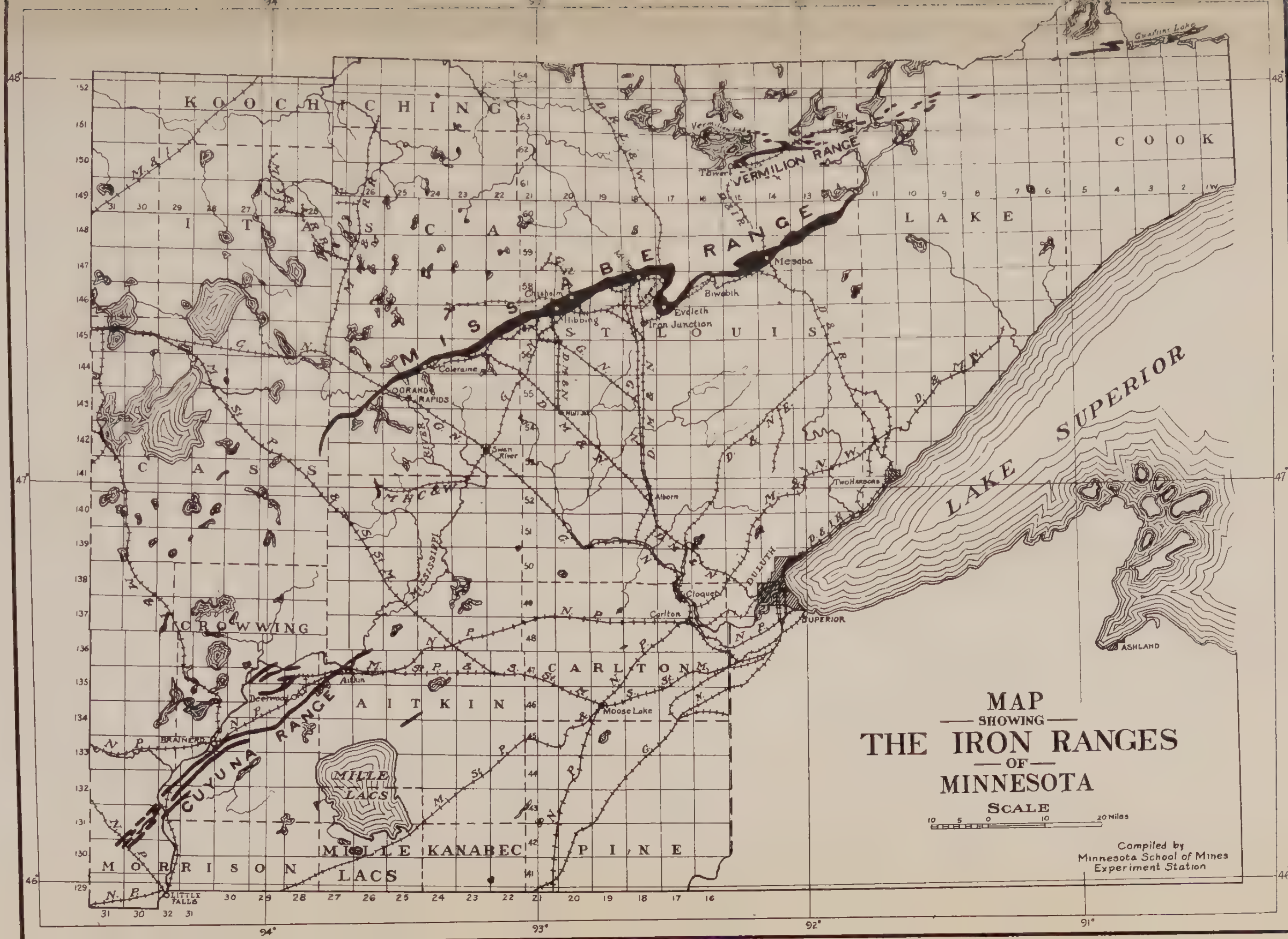
There are three productive iron ranges in Minnesota, the Missabe, the Vermilion, and the Cuyuna. Each range is composed of sedimentary, igneous, and metamorphic rocks of pre-Cambrian Age. The structures are complex, indicating eventful geologic histories. The iron-bearing formations are of sedimentary origin, though there are many exposures of igneous rocks, more or less closely associated with them.

The principal ore formation of the Vermilion Range is of Archean Age. The Missabe ore formation and probably the Cuyuna also, belong to the Upper Huronian. In the Vermilion Range the ore-bearing rocks are closely folded and dip at high angles. Folding in the Cuyuna, although pronounced, is less intense, while in the Missabe gentle folds and low dips prevail.

The portions of the iron formation containing workable ore-bodies have all been subject to weathering or surface decomposition, though visible evidence of this is lacking in some deposits. Surface water is seldom pure; it usually contains in solution small quantities of salts obtained from the atmosphere or the soil. As this water trickles down through the iron formations it takes into solution many elements, especially the alkalies, lime, and magnesia. Thus charged it becomes a solvent for silica. Iron is much less soluble and, therefore, remains behind while the impurities are removed. Wherever fracturing has permitted the free circulation of water, leaching has taken place. Therefore the fractured and rotted portions of the iron formations are invariably much richer than the less permeable portions. In portions of the Vermilion this weathering and concentration preceded the folding of the rocks into the pitching troughs characteristic of this range; therefore the rotted, enriched portions were folded with the poorer portions of the iron ranges so that the workable ores may be found at depths as great as 2,000 feet below the present surface. The Missabe ore formation has not been folded to any extent since it was exposed to surface weathering and concentration; consequently the ores are nearer the surface and no workable ore-body, proved so far, exceeds 500 feet in depth. In the Cuyuna Range workable ore is found at greater depth.

Surface leaching removes the silica, alkalies, lime, and magnesia, concentrates the iron oxides, and leaves the soft and very porous iron ore characteristic of the Missabe. In the Vermilion the deep ores are hard and dense. Some of them, at least, are undoubtedly portions of the iron formation weathered and enriched and later infolded and indurated by pressures exerted in pre-Cambrian times.

[1]Review by W. H. Emmons.


MAP
SHOWING
THE IRON RANGES
OF
MINNESOTA
SCALE
20 Miles
Compiled by
Minnesota School of Mines
Experiment Station
MISSABE RANGE
VERMILION RANGE
CUYUNA RANGE
LAKE SUPERIOR
KOOCHICHING
ITASCA
ST. LOUIS
LAKE
COOK
CASS
CROWWING
AITKIN
CARLTON
PINE
KANABEC
MILLE LACS
MORRISON
Duluth
Superior
Ashland
Two Harbors
Cloquet
Carlton
Moose Lake
Aitkin
Deerwood
Brainerd
Little Falls
Hibbing
Chisholm
Eveleth
Biwabik
Mesaba
Iron Junction
Coleraine
Grand Rapids
Swan River
Alborn
Tower
Ely
Vermilion Lake
Mississippi River

MISSABE RANGE

Location and topography.—The iron deposits of the Missabe are local concentrations in a ferruginous sedimentary formation which extends from a point about 12 miles southwest of Pokegama Lake to Birch Lake, a distance of slightly over 100 miles. The rocks have a strike of about N. 73° E. with a slight dip toward the south. The iron-bearing belt is generally from 1 to 3 miles wide. At Virginia and Eveleth there is a loop in the formation and an increase in width to nearly 9 miles. Running parallel to the iron formation and north of it is a ridge called the Giants or Missabe Range. In the western part of the district this ridge is only 1,400 feet above sea level and not appreciably higher than the surrounding country. From Virginia eastward it is from 1,700 to 1,900 feet above sea level and some 400 feet above the surrounding country. The lower areas are covered with drift, and much of the country is marshy and thickly wooded so that opportunities for areal geological study are limited. Diamond drilling has given much information regarding the distribution of the iron formation.

Rock formations.—The following succession of formations is given by Leith.[1] The younger formation is named first.

Quaternary system	
Pleistocene series	Glacial drift
Unconformity.	
Cretaceous system.	
Unconformity.	
Algonkian system.	
Keweenawan series	Great basal gabbro (Duluth gabbro) and granite (Embarrass granite), intrusive in all older formations.
Unconformity.	
Huronian series.	
Upper Huronian (Animikie group)	Acidic and basic intrusive rocks. Virginia slate. Biwabik formation (iron-bearing). Pokegama quartzite.
Unconformity.	
Lower-Middle Huronian	Giants Range granite, intrusive in lower formation. Slates-graywackes-conglomerate formation.
Unconformity.	
Archean system.	
Laurentian series	Granites and porphyries.
Keewatin series	Greenstone, green schists, and porphyries.

ARCHEAN

Keewatin and Laurentian.—The oldest rocks of the district are Archean greenstones, green schists, and porphyries of the Keewatin series. These are associated with, and probably cut by, Laurentian granites and porphyries. The Archean rocks are exposed here and there in the central portion of the district, chiefly in the vicinity of Iron Mountain, Virginia, Eveleth, and Biwabik. Some, but not all, of the Archean rocks have a well-developed cleavage which is nearly vertical and strikes about parallel to the axis of the range.

[1]Leith, C. K., The Missabe Iron Bearing District of Minnesota, Mon. 43, U. S. Geol. Survey, 1903. Leith, C. K., and Van Hise, C. R., Geology of the Lake Superior Region, Mon. 52, U. S. Geol. Survey, 1911, p. 159.

ALGONKIAN

The rocks of the Algonkian system rest unconformably upon the Archean rocks and are not so greatly metamorphosed. The oldest member of this system is a sedimentary series of conglomerates, graywackes, and slates, of Lower-Middle Huronian Age. The most extensive exposure is from Eveleth eastward to Biwabik. There is a smaller exposure north of Missabe and Trimble.

THE LOWER-MIDDLE HURONIAN BEDS

The Lower-Middle Huronian beds strike approximately with the axis of the range and they dip nearly 90°. Their thickness is probably not over 3,000 or 5,000 feet. These rocks are intruded by the Giants Range granite which usually forms the north boundary of the Missabe district.

UPPER HURONIAN

The Upper Huronian[1] is composed of (1) the Pokegama quartzite, consisting mainly of quartzite but containing also conglomerate at its base; (2) the Biwabik formation, which rests upon the Biwabik and consists of ferruginous cherts, iron ores, slates, greenalite rocks, and carbonate rocks, with a small amount of coarse detrital material at its base; and (3) the Virginia slate. Between the Pokegama quartzite and the Biwabik formation a slight break in deposition is indicated by conglomerate. The Biwabik formation grades into the Virginia slate both vertically and laterally. Some acidic and basic intrusive igneous rocks are associated with the Upper Huronian sediments.

All of these rocks were formed after the close folding which affected the Lower and Middle Huronian sedimentary rocks. Consequently the sedimentary Animikie or Upper Huronian group are not on edge, but generally dip at low angles.

Pokegama quartzite.—The Pokegama quartzite, which is the lowest member of the Animikie group, is of considerable importance, since at many places it forms the base upon which the iron-bearing formation rests. Its thickness varies from a few feet to 200 feet and for so thin a formation it is fairly persistent. From Iron Mountain westward to the end of the iron bearing district it forms a continuous belt from a few feet to half a mile or more in width. East of Iron Mountain it is nearly but not quite continuous to Embarrass Lake. From Embarrass Lake to the eastern extremity of the district it is exposed only here and there.

The iron-bearing formation.—The Biwabik or iron-bearing formation extends along the range for its entire length. Its average thickness is about 800 feet, but, owing to the prevailing low dips, the width exposed varies from one quarter of a mile to three miles. Like the other rocks which generally lie at low altitudes the iron formation usually is covered with glacial drift which varies in thickness from 20 or 50 feet, to 150 to 200 feet. The drift is thicker in the western part of the district than in the eastern part, where the Giants Range is higher. Usually, as already stated, the iron-bearing formation rests upon the Pokegama quartzite, but where that

[1]Loc. cit. p. 163.

is lacking it is in unconformable contact with the older Huronian or the Archean rocks. At the east end of the district the Embarrass granite has been intruded between the iron-bearing formation and the older rocks.

Everywhere along the productive part of the district, the iron formation south of its outcrop is capped by the Virginia slate, but east of Trimble to Birch Lake the Duluth gabbro overlaps the iron-bearing rocks. To the east, the formation is cut off by the Duluth gabbro; on the west it probably thins out, the Pokegama quartzite and Virginia slate coming together.[1] The great bulk of the Biwabik formation is ferruginous chert, with varying amounts of amphibole, some lime and iron carbonate and bands and shots of iron ore. Associated with the chert, mainly in the middle zone, are the iron ores, which occupy about five per cent of the total surface area of the iron-bearing formation. In thin layers and zones throughout the iron-bearing formation, and particularly in its upper horizons, are layers of slate and of paint-rock, the paint-rock usually resulting from the alteration of the slate. Associated with the slaty layers in the iron-bearing formation or closely adjacent to the overlying Virginia slate are green rocks made up of small green granules of ferrous silicate called greenalite.[2] The greenalite has at some places been replaced by cherty quartz, magnetite, hematite, limonite, siderite, and other minerals, and associated with the greenalite rocks are small quantities of lime and iron carbonates.[3]

At the east end of the range near Birch Lake the iron formation has been considerably metamorphosed in consequence of intrusion of granite to the north and of gabbro to the south. As a result considerable amphibole has been developed in the ferruginous rocks, magnetite has segregated into layers, and the rocks have become indurated. As already stated only a small part of the iron formation is iron ore. Certain ferruginous but unpayable portions of the formation are called taconite. The processes which resulted in the concentration of workable deposits are discussed on page 18.

Virginia slate.—The Virginia slate rests above the Biwabik formation and forms the southern boundary of the district. At the east end of the district, however, the slate is not exposed, but the Biwabik is capped by the Duluth gabbro. Near the contacts with the gabbro the mineral composition of the slate is changed and typical heavy silicate minerals are developed.[4] At its base the slate grades into the underlying iron formation which itself is slaty in places. The thickness of the Virginia can not be stated, since the top is not exposed.

CRETACEOUS

Thin beds of conglomerates and shales of Cretaceous Age, lying nearly horizontal, cap the various Algonkian and Archean formations. The basal beds of the Cretaceous locally carry beds of detrital iron ores derived from the weathered iron-ore formation.

[1]Loc. cit. p. 164.
[2]Leith, C. K., The Mesabi Iron-Bearing District of Minnesota. Mon. U. S. Geol. Survey, vol. 42, 1903. Spurr, J. E., Bull. Geol. Nat. Hist. Survey Minnesota No. 10.
[3]Loc. cit. p. 171.
[4]Winchell, N. H., Geol. Nat. Hist. Survey Minnesota, vol. 5, 1900, p. 990. (Final report.)

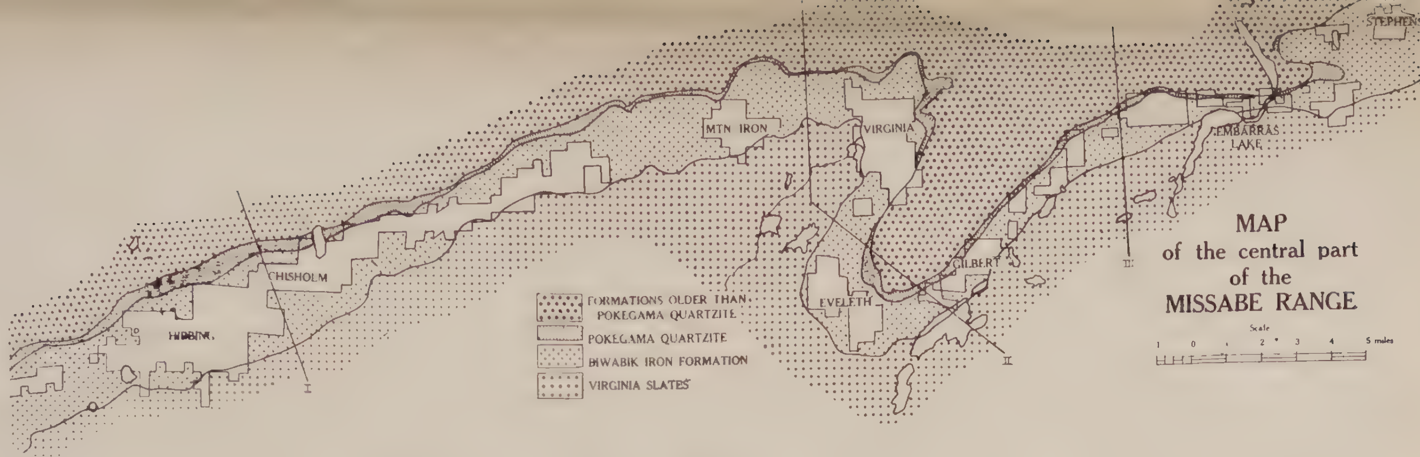


FIG. 2

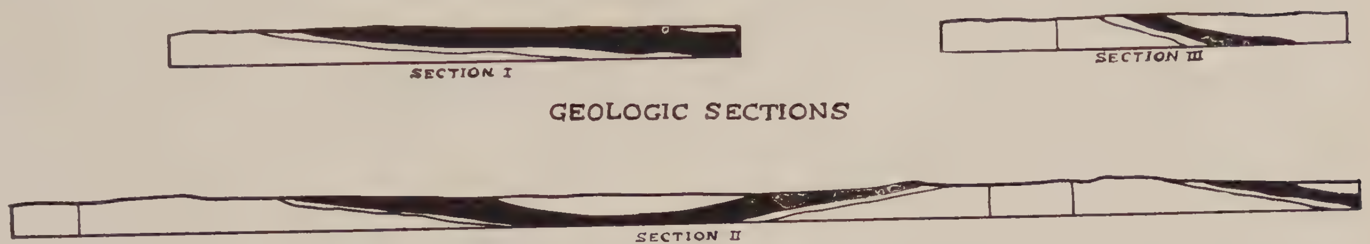


FIG. 3

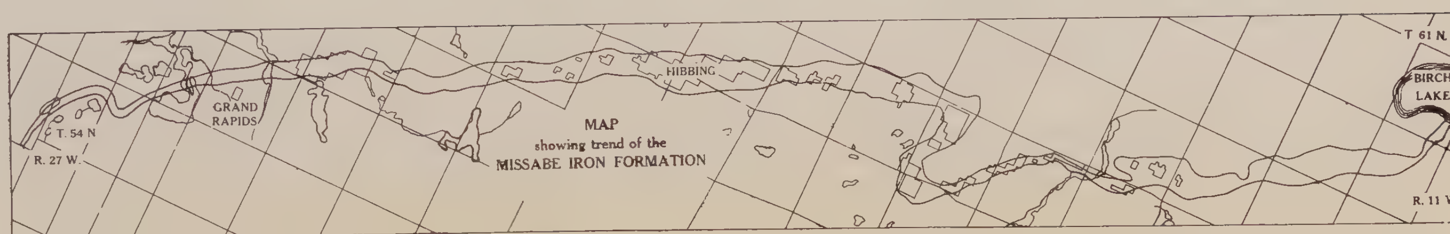


FIG. 4

STRUCTURE

The Middle and Lower Huronian beds and also the Archean formations were subjected to close folding before the Upper Huronian, including the Pokegama, the iron-bearing formation, and the Virginia slate, was deposited. Consequently the dips of the older beds are seldom in accord with the dips of the iron-bearing series, which varies in general from 5° to 20°, toward the south or southeast. In the vicinity of Virginia, Eveleth, and Gilbert the beds are gently folded and make a broad loop, which widens the ore-bearing district to about nine miles. The dips of the beds are generally normal to the contact of the ore formation with underlying rocks. West of Virginia the dips are nearly everywhere toward the south, but at Virginia the beds make a sharp swing, and between Virginia and Eveleth the dip is northwestward. In the Great Western Pit, about a mile south of Virginia, the iron-bearing beds dip west of north 10°. At Eveleth the beds make a sharp turn, swinging back again, and in some of the pits northwest of Eveleth the iron formation dips nearly due west at low angles. East of Eveleth they resume their northeast strike, and in the mines between Eveleth and Gilbert the beds dip nearly southeast. Dips toward the south and southeast prevail from Gilbert to Biwabik, and eastward to the end of the range. This loop and a section across it (sec. II) are shown in Figs. 2 and 3, which are adapted from Plate VIII, Mon. 52, by C. K. Leith, published by the U. S. Geol. Survey.

CHARACTER AND GENESIS OF THE IRON ORE

The iron ores are local concentrations in the iron formation. This formation, as already stated, contains conglomerate and quartzite layers near the base and here and there thin layers of slate or other sedimentary rocks. The ferruginous layers grade laterally into slate bands and upward the formation grades into the Virginia slate which is, as already stated, a sedimentary deposit. The formation as a whole is extensive laterally and it has a comparatively uniform thickness like a bed deposited in water. For these and other reasons it is concluded that the ore formation is a sedimentary rock. This is the conclusion of N. H. and H. V. Winchell,[1] of J. E. Spurr,[2] of C. K. Leith,[3] of Van Hise and Leith.[4]

The iron-bearing formation, however, is not a common type of sedimentary rock and so far as known it can not be matched by any sedimentary rock that is being deposited to-day. For many reasons it is thought that the waters which deposited the iron ores were derived from volcanic centers and that the iron had been leached out of igneous rocks possibly by hot waters and contributed to the sea. "At one time," Winchell says, "a chain of active volcanoes extended across northeastern Minnesota about where the Missabe iron range is found. This was near the shore

[1]Bull. Geol. and Nat. Hist. Survey Minnesota No. 6, 1891, p. 144. Winchell, H. V., The Mesabi Iron Range. Twentieth Ann. Rept. Geol. and Nat. Hist. Survey Minnesota, 1891, 1893, pp. 111-180.

[2]The Iron-Bearing Rocks of the Mesabi Range in Minnesota. Bull. Geol. and Nat. Hist. Survey Minnesota No. 10, 1894, p. 219.

[3]The Mesabi Iron-Bearing District of Minnesota. Mon. U. S. Geol. Survey, vol. 43, 1903, p. 237.

[4]The Geology of the Lake Superior Region, Mon. 52 U. S. Geol. Survey, 1911, p. 169.

line of the Taconic ocean, and was accompanied by land-locked bays and perhaps by fresh-water lakes. Such marginal volcanoes had a chemical effect on the oceanic water, causing the precipitation of silica and probably of iron. Its basic lavas and obsidians were attacked by the hot waters and were converted by encroaching silica into jaspilite."[1]

Van Hise and Leith have shown, moreover, that all of the iron-bearing formations of the Lake Superior district are more or less closely related in time and place to the extravasation of igneous rocks. They have shown also in the laboratory that greenalite and certain associated rocks of the iron formation may be formed under conditions which, it may reasonably be assumed, prevailed at the time the iron-bearing member was deposited.[2]

Only small portions of the Biwabik or iron-bearing formation are rich enough to constitute iron ore. These are patches here and there along the eroded surface of the iron formation, generally not over 200 feet thick, although some are thicker. The workable deposits are secondary concentrations due to the action of surface waters which have leached out the silica and some other elements, and have left the iron in a more highly concentrated deposit. This conclusion was stated by Spurr in 1894. The processes have been studied and their results worked out quantitatively by C. R. Van Hise, C. K. Leith, and Warren J. Mead.[3]

In the Stevenson Mine four samples were taken from the same layer where the ore grades into taconite. No. 1 is the fresh, or but slightly altered, taconite; No. 4, the leached taconite or low grade iron ore; and Nos. 2 and 3 are intermediate, partly altered specimens. The chemical analyses and the measurements of the pore space are given in the table below.

Analyses of ferruginous chert in various stages of alteration.[4]

	Chemical composition					Approximate volume composition			
	Fe	SiO_2	P	Al_2O_3	Loss on ignition	Pore space	Hematite and limonite	Quartz	Kaolin
1.	29.47	52.89	0.016	0.62	2.92	8.00	32.35	57.90	1.74
2.	33.01	50.08	.016	.35	1.65	16.50	31.25	51.40	.93
3.	35.26	43.44	.013	.40	4.48	26.30	33.51	39.30	.92
4.	48.88	25.03	.015	.21	3.83	52.70	30.81	16.18	.34

The greenalite is shown to alter to taconite or to ferruginous chert and the latter to iron ore by the loss of silica. Through the leaching process magnesia, small amounts of lime, and alkalies are also dissolved out and these aid in the solution of silica. The net result is to concentrate the iron and to develop corresponding pore space, though the latter is decreased by slumping of the cellular, porous, weak iron ore. Evidence of such slumping is found in many of the mines where the ore beds dip toward the thicker and richer parts of the ore-body which in general are near the central portions.

[1]Geology of Minnesota, vol. 5, 1900, p. 997.
[2]Loc. cit. p. 519.
[3]Loc. cit. p. 194.
[4]Loc. cit. p. 188.

Since the secondary concentration of the ores has been accomplished by the agency of ground waters, it follows that one is warranted in seeking the workable deposits in the fractured and shattered portions of the iron formation.[1]

Concentration of this nature, where water solutions have found more ready access, has been going on through long geological periods. That it was well advanced in Cretaceous times is shown in the detrital zone of the Cretaceous which carries abundant iron ore in the form of polished pebbles.[2]

THE VERMILION RANGE

Location.—The Vermilion Range is in the northeastern part of the State, in St. Louis, Lake, and Cook Counties. From near the west end of Vermilion Lake it extends about 20° north of east to Gunflint Lake on the Canadian boundary. It is about 100 miles long and varies in width from 5 to 15 miles.

Literature.—The papers treating the geology and ore deposits of the Vermilion Range include Bulletin No. 6, The Iron Ores of Minnesota, by N. H. and H. V. Winchell,[3] Monograph 45,[4] The Vermilion Iron-Bearing District of Minnesota, by J. Morgan Clements, and Monograph 52,[4] The Geology of the Lake Superior Region, by C. R. Van Hise and C. K. Leith.

General geology.—The geology of the Vermilion is not as simple as that of the Missabe. Instead of the gentle tilting or very open folding characteristic of the latter district, the iron-bearing rocks of the Vermilion are characterized by very close folding and a higher degree of metamorphism by pressure.

The iron ores are found in three great geological series, two of which are much older than those of the Missabe Range. The stratigraphic succession as stated by J. Morgan Clements[5] is as follows:

Quaternary system	
Pleistocene series	Drift
Unconformity	
Algonkian system	
Keweenawan series	Duluth gabbro and Logan sills
Unconformity	
Huronian series	
Upper Huronian (Animikie group)	Rove slate Gunflint formation (iron-bearing)
Unconformity	
Lower-Middle Huronian	Intrusive rocks: Granites, granite porphyries, dolerites Knife Lake slate Agawa formation (iron-bearing) Ogishke conglomerate
Unconformity	
Archean system	
Laurentian series	Granite of Basswood Lake and other intrusive rocks
Keewatin series	Soudan formation (iron-bearing) Ely greenstone, a basic igneous and largely volcanic rock

[1]Spurr, J. E., loc. cit. p. 180.
[2]Van Hise and Leith, loc. cit. p. 197.
[3]Geol. and Nat. Hist. Survey Minnesota.
[4]U. S. Geol. Survey.
[5]Mon. 45, U. S. Geol. Survey. Mon. 52, U. S. Geol. Survey, p. 118.

The Ely greenstone which is the oldest is also the most extensive formation in this district. It consists mainly of altered, somewhat basic igneous rocks which were probably in the main surface flows. At many places, though not everywhere, these rocks are highly schistose. The Soudan, which was deposited above the Ely greenstone, is the oldest iron-bearing formation in the district. It is made up chiefly of beds of jasper, and contains also some slates and conglomerates. Some beds are composed largely of iron oxides or iron carbonates. The Ely greenstone and the Soudan formation are intruded by dikes and irregular masses of granite, felsite, and porphyries which locally are metamorphosed to schists. These and the rocks they intrude were deformed and otherwise altered and in places were eroded before the rocks of the next system, the Algonkian, were deposited.

The lower system of Algonkian rocks is termed the Lower and Middle Huronian series. The oldest member of this series is the Ogishke conglomerate, which locally has a thickness of over 1,000 feet. At some places, however, it is only a few feet thick, and at other places it is wanting altogether. Above the Ogishke conglomerate, in the eastern part of the district, is the Agawa formation which contains beds of slate, jasper, iron oxides, and iron carbonates. This formation reaches a thickness of 50 feet and contains some unimportant iron ores.

Above the Agawa is the Knife Lake slate, which is probably several thousand feet thick. It is composed of slates, graywackes, conglomerates, and other rocks. The Knife Lake slate and older formations are intruded by the Giants Range and several other granites, including the Snowbank and the Cacaquabic granite.[1]

The Upper Huronian rocks which contain the iron ores in the Missabe Range are of little importance in the Vermilion district, although they occur in a small area west of Gunflint Lake.

The iron ores.—The iron ores of the Vermilion Range are almost exclusively in the Soudan formation of the Archean. The principal developments are at Ely and at Tower. The Soudan rests upon the Ely greenstone, and is closely and intricately infolded with it. In places the ores are found 2,000 feet below the surface. The ore formation has been subjected to close folding and subsequent erosion has removed the upper parts of many folds. Consequently the remnants of the formation are found at many places in pitching troughs which as a rule are bottomed by the Ely greenstone, or by the soapstone or paint-rock which are in general the altered phases of the Ely, or of certain aluminous or magnesian portions of the Soudan. The paint-rock or soapstone and associated rocks are comparatively impervious to water and thus form a trough-like structure enclosing parts of the Soudan. The Soudan formation consisted originally of cherty iron carbonate and probably banded chert and iron oxide. Surface waters circulating down the tilted troughs leached out the silica and certain other elements, leaving the iron oxide in a porous and more highly concentrated form. Some of this concentration took place before the Lower Huronian sedi-

[1]Mon. 52, U. S. Geol. Survey, p. 136.
Rept. Geol. Survey Minnesota, vol. 4, 1899, pp. 442-448.
The Geology of Kekequabic Lake in Northeastern Minnesota, with Special Reference to an Augite-Soda Granite: Twenty-first Ann. Rept. Geol. and Nat. Hist. Survey Minnesota, 1893, p. 55.

ments were deposited for the conglomerates of the latter contain detrital iron ores. Close folding following the Lower Huronian deposition rendered the ores hard, anhydrous, and crystalline.[1] At Ely much concentration has taken place also after the deposition of the Lower Huronian. Because the ore formations in the Vermilion district were closely folded after some concentration by ground water had taken place, the enriched ores are found locally as deep as 2,000 feet below the present surface.

THE CUYUNA RANGE

The Cuyuna Range[1] is southwest of the Missabe. It extends from the town of Aitkin through Deerwood and Brainerd to a point beyond Fort Ripley. As now defined the iron formation is about 55 miles long with a trend of N. 50° E. and of undetermined width. This range is a comparatively recent discovery and has just entered the producing stage. It was discovered by drilling areas of strong magnetic attraction. The surface is nearly level and the ore formation is overlaid by a deeper drift than exists on the Missabe.[2]

The rocks of the district as exposed by diamond drilling are mainly slates and ferruginous carbonates. Prof. N. H. Winchell[3] regards them of Archean Age like the ores of the Vermilion Range, but Dr. Leith and others have referred them to the Upper Huronian so extensively developed in the Missabe Range. The succession is stated to be as follows:

Quaternary system	
Pleistocene series	Glacial drift, 35 to 400 feet thick
Cretaceous system	Sediments, small areas
Algonkian system:	
Keweenawan (?) series	Igneous rocks, extrusive and intrusive, basic and acidic
Huronian series:	
Upper Huronian (Animikie group)	Virginia slate: Chloritic and carbonaceous slates with small amounts of interbedded graywackes, quartzite, and limestone. Thickness great. Where intruded by igneous rocks, garnet is developed.
	Deerwood, iron-bearing member of Virginia slate, contained originally much iron carbonate, but is largely altered to amphibole-magnetite rocks, ferruginous slate and chert, and iron ore. It is found in lenses in the Virginia slate, presumably near the base.

The sedimentary rocks strike in general about N. 50° E. and are folded closely so that they dip at high angles. At some places the quartzites and slates are changed to

[1]Loc. cit. p. 142.

[2]Leith, C. K., The Geology of the Cuyuna Iron Range Minnesota. Econ. Geology, vol. 2, 1907, pp. 145-152.

Zapffe, Carl, The Cuyuna Iron-Ore District of Minnesota. Supplement to the Brainerd (Minn.) Tribune, 1910, pp. 32-35. (With map.)

Adams, F. S., The Iron Formation of the Cuyuna Range. Econ. Geology, vol. 5, 1910, pp. 729-40; vol. 6, 1911, pp. 60-70, 156-180.

Leith, C. K., and Van Hise, C. R., Geology of the Lake Superior Region. Mon. 52, U. S. Geol. Survey, 1911, p. 211.

[3]The Iron Ranges of Minnesota and Their Differences. Bull. Minnesota Acad. Sci., vol. 5, No. 1, 1911.

schists. Toward the south and west where intrusive rocks are abundant these slates become garnetiferous, staurolitic, and hornblendic.[1]

The iron-bearing member, which was originally an iron carbonate rock, is intruded by acid and basic igneous rocks and locally has been changed to an amphibole-magnetite rock. This rock, because of induration attending the development of the silicate minerals, is not leached of silica as readily as iron carbonate rocks and consequently is not so favorable a place for the development of the secondary richer iron ores. The workable deposits are vertical or steeply dipping lenses which have a maximum width of about 400 or 500 feet. Their average depth is about 300 feet, but the maximum known depth is 850 feet. Some are developed a half a mile along the strike. The ores, soft and hard, are in the main non-Bessemer, and some contain considerable manganese. The soft ores include a powdery, high-grade, brown ore, containing 55 to 63 per cent iron, and a lean, reddish-purple ore containing 45 to 50 per cent iron. Most of the hard ore is dark brown hydrated hematite. It is closely stratified and has suffered brecciation as a result of slumping, following the leaching out of silica. As a rule it carries from 50 to 60 per cent iron.[2]

[1]Hall, C. W., Keewatin Area of Eastern and Central Minnesota. Bull. Geol. Soc. America, vol. 12, 1901, pp. 343-376, quoted in Mon. 52, p. 213.
[2]Loc. cit. p. 219.

MISSABE PROSPECTING

Few people realize the difficulties encountered by the early prospectors on the Missabe. The country was a dense, almost trackless forest, alternating with vast swamps. Distances were great, and, while water and timber were always at hand, food and other supplies the prospector had to pack in on his back. Surface indications of ore were almost wholly absent. While some of the earliest valuable discoveries were made by chance, such as the finding of ore under the roots of fallen trees on the Biwabik and Hale Mines, the tracing of the Missabe formations and the discovery of valuable ore-bodies is in the main the result of careful study and resolute enterprise, intelligently directed. Early exploration was practically limited to test pitting down to the water line. The understanding of geological conditions resulting from the rapid development of the earlier discoveries quickly led to the use of churn and diamond drill prospecting and to the realization that the drill was a reliable and efficient prospecting tool. Present-day methods, the result of fifteen years' experimentation, are remarkably rapid, efficient, and, comparatively speaking, cheap. A thousand or more men are constantly engaged under competent engineering supervision in exploration work on the Missabe alone and the annual expenditure for this work must approximate two million dollars.

The unit of exploration is a 40-acre tract. Work is begun by running out the boundary lines of the "forty." These are usually quite irregular. Five holes are then put down, spaced about 500 feet apart. If no ore is found, the ground is temporarily abandoned. If ore is found, the tract is surveyed, 100-foot coördinates are run, the topography is taken, and supplementary drilling is done on the intersections, the practice varying with local conditions from 100- to 300-foot intersections. Drilling is continued in each hole until taconite is struck. If this occurs close to the surface, the hole is continued a reasonable depth, since many valuable ore-bodies are found under quite a depth of taconite. What this distance shall be is a matter of judgment, to be determined for each case. One hole in each tract is drilled through the iron formation to quartzite.

A typical drilling camp on the western Missabe was visited and studied to furnish a general description of equipment and methods. Operating details and costs were gathered from various sources and are not to be applied especially to this camp. The camp consists of ten buildings including cook-house and bunk-house for, roughly, 120 men. Weather conditions compel a fair class of construction, however rough and temporary the buildings may be. Within a radius of three or four miles of this camp from 20 to 40 drills were at work. At the time of inspection 30 drills were in operation. The crew was divided as follows: Thirty drill runners and sixty

helpers; five pump-men, one blacksmith, three diamond setters; two cooks, four stewards and waiters; one superintendent, one clerk, two foremen, three samplers. Each crew has a complete outfit for both churn and diamond drilling. The churn drill is used to drill through surface, soft ore, soft slate, and other soft strata. The diamond drill is used when drilling in taconite, hard slate, hard ore, and quartzite—in short, whenever the churn drill can not make satisfactory headway.

A typical Missabe outfit costs about $2,500, exclusive of diamond investment. It consists of:

Sullivan diamond drill "H," 1,000-ft. capacity.

No. 5 Cameron pump.

Evered model churn drill.

Vertical dry-top boiler, 20 h. p. (Extra heavy pattern on account of the hard usage to which they are subjected.)

String of tools, drill rods, 200 feet of 3-inch and 500 feet of 2-inch casing.

The tripod or derrick is about 30 feet high and consists of four 6- to 10-inch round poles, two of them being made into a ladder by nailing on inch planks to serve as steps. These timbers are all bolted together at the top of the derrick and an 18-inch solid iron head pulley is used. The weight of the outfit is, roughly, 8,000 pounds, exclusive of equipment.

The churn drill used on the Missabe is a local development manufactured in Duluth. A perforated, hollow cutting bit is screwed into the bottom of a jointed hollow rod, the upper end of which is connected by a flexible coupling to a No. 5 Cameron force pump. This cutting tool is hung from a rope that passes over a pulley at the top of the derrick to a small drum geared to an 8 to 10 h. p. steam engine. This end of the rope is left loose, and the drilling or cutting action is obtained by the driller alternately tightening and easing off on this rope which passes several times around the drum. The drill pump forces water down the hollow rod and out through the perforations in the chisel bit. The hole is cased and the drill water is under sufficient pressure to carry the sludge to the surface. The stroke of the drill is from 5 to 8 inches and the drill must be turned at each stroke.

Casing is kept close to the bottom and is driven with a 200- to 300-pound hammer. Casing is likely to stick fast in the hole unless it is turned continually while it is being driven. In surface drift extra heavy 3-inch lap-welded pipe is used for casing. If necessary to case below, the size is reduced to 2 inches. The smaller casing is not strong enough to stand the pounding through the drift. Casing comes in lengths of 18 to 21 feet and is threaded on the job. Three-inch casing lasts about one year in continuous service and may need re-threading several times. Two sizes of churn drill rods are used. Extra heavy 1¼-inch pipe for surface and 1-inch pipe for all subsequent work. The churn rope is ordinary 1¼ inch hemp. It lasts from one to two months, at which time it becomes unsafe to hoist diamond drill rods. The breaking of a rope and dropping of a line of rods might break the diamonds with a consequent heavy loss.

For diamond drill work the Sullivan "H," capable of drilling a 1,000-foot hole, is largely used on the range. The hydraulic feed type, while requiring a more experienced and careful operator, gives the best results and seems to be in favor; about 80 pounds pressure is required. A drill rod of 1⅛-inch outside diameter is used in 10-foot sections. Short sections 3 feet long are used as the bit advances, two of these being replaced by a regular 10-foot section. The core barrel varies from 10 feet in soft, to 5 feet in hard, rock. The drill rods are usually raised and lowered in 20-foot lengths.

OPERATING DETAILS

Setting-up.—Two heavy items of expense are road building and water piping. Each drill crew, consisting preferably of three men, builds its own roads, filling low places with logs, bridging streams, and leveling off uneven places. Old logging roads

Fig. 5. Missabe Prospecting Rig.

are used whenever possible to minimize the expense on new work. It usually takes a drill crew two to four days to move; this includes tearing down, building roads, loading, haulage, unloading, and setting up at destination. It takes from three to ten hours to set up and start, the length of time depending upon the amount of water pipe to be laid. Four wagons are in constant use in this camp for moving rigs and hauling coal. Two or three teams are employed snaking logs. Four wagons usually move an outfit. The boiler is the hardest to move as four horses are required. In swampy ground wagon roads are either too expensive or impossible and everything is

moved by hand. The boiler fittings are then removed and the boiler is rolled to its new location.

Drills need quite a volume of water. When there is a stream or spring, the pump suction is simply lowered into it. Again when water is less than 500 feet away, an injector may be used to pump water to the boiler. A steam line is run from boiler to injector and the water line parallels it. This is satisfactory for low heads. Often pump stations must be established to pump water to a number of crews. At this camp there are five or six such stations equipped with 42x84-inch boiler and No. 5 Cameron pump. These pumps supply a 2-inch main with 1-inch branches. The drill crew dig a water hole beside their drill 3 to 4 feet deep, 8 to 10 feet in diameter. The pump draws its water direct from this hole. The return drill water runs back into this hole, the sediment having been precipitated in the sampling barrels. During freezing weather the supply pipes are raised 6 to 8 feet above the ground and are given sufficient slope so that they will drain off when the pump stops. In severe weather the water supply often gives trouble. During cold weather the drill rig is covered in to a height of 10 feet by a portable housing. The house fits around the tripod and has holes in the roof for rope and drill rods. This housing is made of inch plank and corrugated siding and roofing on 2x4 studding. Fig. 5 shows two views of a typical Missabe drilling outfit.

Churn drilling.—Surface drilling through glacial drift is subject to many difficulties. Casing is driven with a 200- to 300-pound weight, consisting often of a 10-inch cast iron cylinder with a 2-inch hole through the center, through which an ordinary drill rod may pass to serve as guide to the falling weight. The casing must be constantly turned during driving to prevent it from sticking fast. The drift is full of boulders of all sizes; these must be blasted. The ordinary procedure is to use from one to six 8 inch sticks of ⅞-inch 60 per cent Hercules dynamite, wire them together, insert an electric exploder and lower this into the hole by means of the electric lead wires connected to a blasting machine of the push-rod type. The casing is raised out of the way and the blast exploded. Large boulders often need several blasts and sometimes it takes a day or two to pass through one boulder.

Driving casing through quicksand (with or without boulders) is often slow and difficult work. The quicksand settles and packs around the casing which sometimes sticks as solidly as though set in cement and can not be budged. The wash-pipe may be used to wash out a big hole; if there are any boulders in the sand, they are apt to crowd in and settle around the pipe; this trouble is aggravated when the boulders are small. Blasting casing, while seemingly a simple operation, is at times subject to unaccountable troubles. Cap after cap may go off, failing to explode the charge; sometimes days at a time are consumed in vain attempts. The bottom of the casing may remain intact and unmoved while the casing may split longitudinally 25 to 100 feet above the point of the blast. When a diamond drill hole in hard rock strikes ore of greater thickness than 5 feet, it must be enlarged in order to case the hole through with the regular 2-inch casing. This is locally known as "blasting down casing" and is subject to some variation in practice. A simple and common way is to

lower a long string of dynamite from 10 to 20 feet in length to the bottom of the hole, explode it, and wash up as much of the shattered material as possible with a wash pipe. This process is continued from the bottom upward until the point is reached where casing originally ceased. The casing is, of course, hoisted sufficiently to be out of danger. The hole is then chopped out and casing driven in the usual way.

The charge consists of a number of 8-inch sticks of 7/8-inch dynamite. The sticks are wired together so that they will hang straight in the hole, the exploder is inserted in the upper stick, and the reinforcing wires are attached to the electric lead wires. The reinforcing wires are sometimes replaced by a tin cylinder, the exploder being placed in the upper stick; the cylinder is lowered into the hole by the electric lead wires. Again the charge may be placed in a rubber tube, reinforced with wire as before described.

If there is a strong natural flow of water due to an underground watercourse or spring, the charge must be weighted down to sink it. For this purpose ready-made 3/4-inch lead pipe cartridges packed with 75 per cent dynamite are used. This lead blasting pipe is purchased from the Du Pont Powder Company at 20 cents a foot; the driller simply cuts off a 5- to 20-foot length, which he carefully primes and lowers down the hole by the blasting wires. The lead is soft and makes no trouble if a piece should remain in the hole. Special electric caps, costing $1.25 each, are used for this work. Blasting casing is subject to many mishaps and is apt to be very slow work. One crew of three men, working day shift only, drilled 50 feet in taconite, blasted down and cased the entire distance with 3-inch casing in one week (6 days), whereas another crew took over 3 weeks for the same work, most of the extra time being consumed in blasting down the casing.

Sampling.—The surface drift is not sampled. Three-inch casing pipe is used through the surface, sand, soft slate, etc., to ore or rock. In case sand overlies the ore this casing is driven a few feet into the ore to prevent mixing sand with the sample which is carefully taken out with the churn bit, followed closely by a 2-inch casing. When hard rock underlies the drift, the churn is replaced by a diamond drill and a 1½-inch hole continued through the rock. Should ore be struck, this is penetrated 5 feet with a chopping bit. If still in ore, the rule is to blast down the casing to the ore and continue the sampling through the ore with the usual churn bit followed closely by casing. This practice is sometimes varied in new territory about which little is known or in passing through alternate layers of slate and ore by continuing the 3-inch casing as far as possible. Ore samples are taken at 5-foot intervals. Water is usually pumped into the hole while the drill is running and the cuttings are washed out continuously. This is always done when the drill is passing through rock and ore samples also are usually taken in this manner. Under certain conditions more accurate results can be obtained in sampling by drilling dry, that is, stopping the pump, drilling 5 feet with just the necessary water (or ground water), driving the casing the full five feet, and rechopping any material that may pack during this driving. The sample is subsequently washed out in one operation with the full force of the wash pump. The drill water and cuttings are flushed out through a T at the top of the

casing into settling tanks or barrels. From four to seven barrels are used. In the side of the barrel, 8 inches from the bottom, is a hole fitted with a plug. When the sludge has settled sufficiently this plug is withdrawn and the bulk of the water is run off. The remainder settles in from two to three hours, after which time the water is slowly decanted off. The residue, representing a 5-foot sample, is carefully gathered, dried, mixed, and cut down, the remaining pulp being cut into the requisite number of samples to supply the fee owner, the operating company, the laboratory, etc.

In hard ore, hard slate, quartzite, etc., core samples are taken. If it is found impossible to extract 18 inches of core in 5-feet advance, the sludge is sampled also. In most hard slates and taconite from 2 to 3 feet of core is pulled per 5 feet drilled. While the diamond drill gives an ideal sample in hard rock, in soft ore the churn drill gives by far the best sample. The churn drill sludge is coarser and comes out at a higher elevation above the ground; it settles out better than the finer diamond drill sludge which comes out nearer the collar of the hole. With care, very good samples are, on the whole, obtained. Careful drilling and careful settling of the drill sludge and collection of the sample are essential. In a clean ore-body the drill samples and the subsequent mine samples check surprisingly closely. Streaky ores, having alternate layers of hard material and small streaks of soft ore, show much variation, sometimes as much as 6 to 8 points. The very soft, fine ore often stays in suspension and the water comes up muddy. In the sandy ores of the Western Missabe concentration of values is a general trouble and drill returns may overrun several points. Occasionally the reverse is found. These holes must be drilled very carefully and with the minimum water.

Diamond drilling.—Diamond drilling on the Missabe differs from the standard in this respect only, namely that the diamond drill is used principally as an adjunct to the churn drill. The diamond drill puts down a small hole through taconite quickly and cheaply as compared with the churn drill, which makes very little impression on taconite. If ore is found, the combined cost of drilling the small hole and enlarging it by blasting is less than the cost of drilling a full-size hole with diamonds. If results are negative and the hole is abandoned, the saving is obvious.

The bits used are soft steel 1½-inch outside diameter. They are usually set with 6 stones to give a $^{15}/_{16}$-inch core. The large drilling concerns have a heavy carbon investment. Stones range from 4 to 9 carats; medium stones from 4 to 5 carats are preferred. The price of carbons has gone up to $90 for choice stones. Borts are used to some extent for comparatively soft drilling; they can not, however, be depended on in shattered taconite. The bits are set in the usual way, the larger stones on the outer circumference. The larger stones sometimes project from both the inner and outer circumference of the bit. Stones set without copper packing give better service. The minimum clearance is usually aimed at. The wages drawn by diamond setters amount to $118 a month and a competent man sets from 5 to 6 bits in a 10-hour day; this includes the labor of cutting out the stones from the old bits and resetting them in new ones. In hard, seamy rocks two bits are usually needed per drill shift, one in the morning and a fresh one at noon. Even then considerable ream-

ing out of the hole is necessary before the new bit can safely be forced to the bottom of the hole. The lower third and sometimes the lower half of the distance drilled by the old bit must generally be reamed out before the drill takes hold. Reaming consumes on an average 1½ hours of the drill shift.

Most of the drill work is shallow, only a small proportion of the holes reach 500 feet. The difficulties attendant on deep drilling, therefore, do not obtain. Occasionally a drill rod parts; this usually means a fishing job with special fishing tools. The same applies to casing. If casing breaks in the middle of a section, a small piece of pipe with a cloth wrapped around it is lowered down the hole inside the casing. Gravel and sand are then pumped down until a tight joint is made between the cloth-wrapped pipe and the casing. This method will generally permit the pulling up of the casing. Diamonds rarely fall out of their settings when the proper care is exercised. When they do, they are regained in the usual way with a wax-filled impression cup. Bits are sometimes dropped in the hole. The Missabe work is so shallow that the permanent loss of a bit is of rare occurrence. Such losses frequently occur in deep drilling. The time required to fish out a bit may range from an hour or two to several weeks or months in the hands of even the most expert drillers.

The following log of a Vermilion Range hole, drilled at an angle of 85° through greenstone, jasper, etc., illustrates such a case. After the ninth day the work was run on two 10-hour shifts.

July 20-23. 22 feet in surface and casing.

July 24. 10 feet D. D. core bit and 10 feet with casing bit.

July 26-27. 29 feet D. D. core bit.

July 28. 7 feet D. D. core bit and 21 feet with casing bit.

Total depths, 68 feet core bit, and 31 feet casing bit.

July 29. 30 feet D. D. core bit through greenstone.

July 30. Double shifts put on.

July 30. 20 feet D. D. core bit and 20 feet with casing bit through greenstone and paint-rock.

July 31. 20 feet D. D. core bit and 10 feet with casing bit through jasper and paint-rock.

August 2-7. 124 feet and 95 feet (6 double shifts). Total depth, 426 feet.

August 9-14. 141 feet and 82 feet (6 double shifts). Total depth, 610 feet.

August 16. No work.

August 17-20. 44 feet and 26 feet (3 double shifts). Total depth, 680 feet. Decided to case the hole.

August 21. Getting ready to case with 1⅝-inch.

August 24. Casing from 76 feet to 139 feet.

August 25. Casing from 117 feet to 369 feet.

August 30. Hole cased to 658 feet. Drilling resumed.

September 1. 8 feet and 10 feet. Dip of hole found to be 75°.

September 2-4. 28 feet and 16 feet (3 double shifts). Total, 754 feet.

September 7-9. 33 feet advance and then lost bit in hole.

September 10. Started a 2½-inch casing bit. On October 25th, at a depth of 630 feet this bit was also lost in the hole and work was abandoned after about $3,600 worth of diamonds had been planted in the two attempts.

The log of a more successful hole drilled from an underground position in the same locality shows 1,633 feet drilled through greenstone with a 15/16-inch core in 131 actual shifts, during 94 calendar days, at a cost of about $3.25 a foot, of which about $0.80 represents labor and $2.45 supplies. The diamond wear was practically $1.85 per foot, 43½ carats valued at $3,000 being included in the supplies.

To illustrate the vicissitudes of diamond drill contracting the log of the recovery of a line of rods and a diamond bit stuck for years in the bottom of a 1,730-foot hole is furnished by a well-known Missabe drill contractor.

October 6. Moved off old drill and rigged up "C" drill.

October 7. Washed out hole.

October 8-9. Tried to square up rods and take impression of broken rod in drill hole.

October 11. Made repairs and made special tools.

October 12-13. Drilled 4 inches off top of rod and took impression. Drilled 2 inches more and took impression.

October 14. Took measurements with left-hand rods and put them down with a spear.

October 15. Got tap into rods 2 inches; pulled hard but could not budge them. Turned rod as much as seemed safe. Awaiting orders.

October 16. Repeated above. Broke spear while trying to back off.

October 18. Drove broken spear 6 inches farther.

October 19. Tried to make hole in rod larger with reamer. Apparatus broke.

October 20-21. Tried to get cuttings up. Drilled off 4 inches from top of rod.

October 22. Drilled 6 inches off the core-shell cut in two during the last run.

October 24-25. Fishing for file bit.

October 26. Drilled through file bit.

October 27-28. Tried with spear and taps to get file bit.

October 30. Finished drilling over file bit and brought up pieces of lost bit as core.

November 1-3. Picking out pieces of iron, locating end of rod and squaring it off.

November 4. Taking impression and taking left-hand outside tap.

November 5. Put down tap and broke it trying to back off rods.

November 6. Decided to ream out hole. Getting ready.

November 7-22. Reamed to 1,419 feet.

November 23. Reamed 6 inches over old rods.

November 24-27. Broke down; repaired; reamed 6 inches more.

November 29. Lowered right-hand rods, tapped broken rods, and lowered left-hand rods with bell-tap.

November 30. Lowered a coupling and pounded on broken rod, lowered hollow tap. It was too big. Could not make headway.

December 1. Trying to recover the rod. Used a tripod pole 43 feet long.

December 2-3. Engine and boiler work. Lowered left-hand rods and recovered 120 feet of rods.

December 4. Lowered coupling; could not connect. Ran file bit. Lots of borings and pieces of iron in the way.

December 6-7. Lowered rods with bees-wax. Cleaned out filings and cuttings.

December 8. Fishing.

December 9. Ran slowly. Much iron in hole.

December 10-11. Pulling up half a day. Tried to get iron out.

December 13-14. Hauled "E" drill and rods and "B" drill on to the ground. Set up "B" drill on new foundation.

December 15. Lowered "B" rods in 10-foot lengths. In pulling out the rods blocked. Lowered rods to bottom. Left rods in hole caught by the iron; could not pull them out. Reamed 21 feet.

December 16. Rods caught 4 hours, pulled out and lowered new bit.

December 17. Lowered bit to bottom, left rods in hole. Reamed 50 feet to 1,603 feet.

December 18. Rods caught tight by iron borings, 5 hours getting out.

December 20. Lowered tube bit and reamed 1 foot over "E" rods, left rods down. Drilled 4 inches with tube bit, brought up big pieces of iron, lowered rods, connected up and *pulled out the lost bit.*

December 21. Broke "E" rods, repair work. Reamed 36 feet.

December 22. Started to pull up and broke tripod leg. Repaired, lowered down. Left rods down. Reamed 50 feet.

December 23. Reamed 40 feet to 1,729 feet in quartz.

December 24. Broke pinion tooth and delivered hole to company to finish, the lost bit having been recovered.

Two crews working 2 ten-hour shifts daily, from October 6th to December 24th. Bill rendered, $2,225.

Carbon wear.—The carbon wear in Missabe drilling of course varies between wide limits. Stones are used until they are reduced to ⅛ carat and even less. On ordinary slates the wear is very slight and on a 50-foot run through soft slate the wear would be inappreciable. In hard taconite an average of $2 per foot is reported. On badly seamed taconite it may be twice this. A bit sometimes comes out of a hole showing more wear after a few hours' service in hard shattered rock than a month's regular work would give it. An average of the data available on this subject would give an average wear of $1.50 per foot covering several thousand feet under all conditions. (Price of carbons assumed $90 per carat.)

SPEED AND COST OF DRILLING

The speed of drilling varies greatly. The following speeds may be taken as general averages which do not allow for extraordinary delays and accidents:

Churn drilling.—Churn drill speeds in surface drift range from 25 to 40 feet per shift for depths not to exceed 75 feet. At a depth of 100 feet an advance of 20 feet per shift would be considered good work. Under particularly favorable conditions as much as 120 feet has been made in a shift. In glacial drift, 10 to 100 feet, depending on boulders. In soft slate, 10 to 50 feet per shift. In ore, 10 to 15 feet per shift.

Diamond drilling.—Diamond drill speeds range as follows: In hard slate, 8 to 20 feet per shift. In quartzite, 5 to 15 feet per shift. In hard taconite, 6 to 10 feet per shift. In decomposed taconite, 5 to 15 feet per shift.

The average footage per drill varies considerably with the locality, the experience of the management and operators, the number of drills in operation, and the average depth of hole. With a few exceptions, the average footage per drill (mixed churn and diamond drilling) may safely be placed at 8 to 10 feet per drill shift, allowing for all time lost in break-downs, moving, and other delays.

Cost of drilling.—The operating expense of a drill is about $12 per shift. This includes labor, direct and indirect, various supplies including fuel, which is roughly 400 lbs. of coal per shift, costing about $4 per ton delivered at the camp. To this must be added diamond setting and diamond wear for any portion of the hole so drilled, casing used, wear and tear on machinery, interest on investment, engineering and other "overhead expenses." Some companies have quite an organization, which adds greatly to the value of the results and, incidentally, to the expense.

The actual cost of churn drilling may be stated at $1.75 to $2 per foot, with $3 to $3.50 per foot for diamond drilling. It must not be imagined, however, that the individual mine operator or explorer could attain these results on the basis of a few isolated holes. The prevailing prices of $3 to $6 respectively, while they undoubtedly yield a handsome profit, are quite reasonable when it is considered that this service is available anywhere on the ranges without personal inconvenience, preparation, or responsibility on the part of the recipient. The recipient is assured of unquestioned integrity, of results which are accepted in the open market, and of a guarantee that, no matter what may develop, the hole will be put down at the contract price.

Drill work to be effective must be directed by one who has made an intelligent study of local conditions, one who appreciates the possibility of variation from conditions hitherto found. The following logs illustrate the extremes of Missabe conditions:

Log A. A typical case without difficulties and delays. 168 feet advance in 14 shifts. Three-man crew working day shift only during month of February.

1 day to move and set up.
1 day to pass through 21 feet of surface.
3 days to sample 55 feet of ore.
1 day to sample 12 feet of ore and taconite.
7 days to pass through 80 feet of taconite.
1 day to pull casing.

Log B. Presents the other extreme. A very bad hole giving much trouble. 307 feet in 187 shifts.

2 days moving, setting up, pipe line, etc.

25 days to pass through 148 feet of surface, progress from 6 inches to 17 feet daily.

20 days to pass through 85 feet of taconite with diamond drill.

2 days blasting diamond drill hole and getting casing down from 148 to 166 feet.

9 days blasting diamond drill hole and getting casing down from 166 to 170 feet.

14 days blasting diamond drill hole and getting casing down from 170 to 187 feet.

14 days casings finally pulled out and hole re-cased.

18 days getting casings down 197 feet to 237 feet.

60 days consumed in attempts to case down. The casing was wrecked and pulled out and the hole was re-cased 5 times before the ore was finally reached at a depth of 257 feet.

6 days passing through 17 feet of ore.

6 days passing through 37 feet of ore into quartzite.

Total time consumed 187 shifts.

ORE ESTIMATING PRACTICE

The unit for both exploration and ore estimating purposes is generally considered to be the 40-acre tract. Ore estimates are based primarily on the exploration data (drill holes and test pits) and secondarily upon such supplementary information as may be available from open-pit or underground development work. Estimates fall into various classes. A total tonnage estimate is usually made first. This may later be supplemented if occasion arises by various special estimates such as Bessemer, non-Bessemer, steam-shovel ore vs. underground ore, etc. To make an estimate each property must be regarded as an individual problem; certain recognized principles must be adapted to the particular case in hand. The detailed estimate presented in Plate I will be more comprehensible if preceded by the following statements and definitions:

A sample to be representative should be taken across the formation. Therefore, in these flat Missabe ore-bodies, composed of a number of layers of large area and relatively small thickness, information on vertical lines is of paramount importance. When a piece of ground has been properly explored, drill holes in the ore-body are usually 200 feet apart. There are, however, many exceptions to this, especially in the older prospecting work; the discovery of ore in one hole was often supplemented by other holes variously spaced to conform to someone's theory. On the eastern and central Missabe all ore above 49 per cent is considered merchantable. Material running from 39 per cent to 49 per cent is at present unmerchantable; it is mined and stock-piled as third grade, or ore-material. On the western Missabe this latter classification includes material ranging from 35 per cent to 53 per cent in iron. While small quantities of material running 30 per cent or under may have been treated in special cases, for present purposes the minimum may be placed at 35 per cent. The availability of this material is wholly dependent upon whether it can be washed or concentrated. This class is treated separately on page 38.

Missabe ores as they exist in nature contain a varying quantity of water. The exact percentage of moisture is found for each sample dried at 212° F. The analysis is always made on the dried sample and reported as a certain percentage of iron "dried." All the average grades shown in the recapitulation in the Ore Estimate, Fig. 9, Plate I, are thus reported. Obviously the percentage of iron in the ore in its natural state (called iron "natural") will be proportionately less and a correction must be made as follows:

$$\% \text{ iron "natural"} = \% \text{ iron "dried"} \times (100\% - \% \text{ moisture}).$$

For convenience in calculating all values and measurements are reduced to certain units. The *Foot-Unit* is the product of the value (percentage of iron for instance)

and the depth of ore in feet represented by the sample. A 5-foot sample of 60 per cent ore would therefore be represented by 300 foot-units. The foot-units are computed for each grade at each hole in which ore was found. Thus the aggregate foot-units of each grade may be computed for the entire drilling. Finally, dividing the aggregate foot-units of each grade by the aggregate number of feet sampled, will give the average analysis for each grade in the ore-body. The *Ton-Unit* is the product of tonnage, and value per ton. The total number of tons of each grade multiplied by its average per cent will give the ton-units for that grade. The sum of the ton-units of all grades divided by the total tonnage gives the average grade of ore throughout the ore-body.

The term *Bessemer Ore* is not subject to exact definition. Rukard Hurd defines it as follows: "Bessemer ore dried at 212° F. has a typical analysis of 61.55 per cent iron, 0.045 per cent phosphorus, 4.6 per cent silica, and 1.5 per cent manganese. With a generally accepted moisture of 10 per cent, this is equivalent to 55.39 per cent natural iron. The percentage of iron may be diminished provided there is a diminution of phosphorus equal to 0.000818 for each per cent of iron loss." In short, it might be stated that the fundamental requirements for a Bessemer ore under present market conditions is that the proportion of phosphorus to iron in the ore shall not exceed the equivalent of the following ratio: dried phosphorus 0.045 per cent to natural iron 55 per cent.

The volume of Missabe ore varies from 10.5 to 17 cu. ft. per ton. The volume does not bear a definite ratio to the iron content. The variation is largely due to structural features. There are instances of both a 52 per cent and a 60 per cent ore measuring 15 cu. ft. to the ton, while in another case a 60 per cent ore measures 10.5 cu. ft. to the ton.

TYPICAL MISSABE ORE GRADES

	Per Cent.					
	Iron	Phosphorus	Silica	Manganese	Alumina	Moisture
Bessemer ore	66.52	.023	2.42	0.51	0.30	9.00
	58.20	.043	4.60	2.40	1.20	11.00
Non-Bessemer	63.00	.060	1.80	1.80	2.40	11.20
	53.50	.110	3.33	0.54	4.20	10.87
Western Missabe Washable ore (Ore material)	36.78	.085	39.58	0.60	0.46	10.00
	38.15	.028	35.00	0.55	1.16	10.00
	42.25	.033	31.90	0.25	0.42	10.00
	48.94	.207	16.36	....	0.60	10.00
	53.94	.054	24.86	....	1.77	10.00

The first step in making an estimate is to study and properly map the exploration data. Maps for operating purposes are usually made to a scale of 50 feet = 1 inch, or 100 feet = 1 inch. For estimating and filing purposes a scale of 200 feet = 1 inch is often used. After a plat is prepared showing the location and elevation of drill holes, test-pits, drifts, etc., cross-sections may be made of the ore-body on a vertical and horizontal scale of 40 feet = 1 inch. Cross-sections are usually made to cover one "40" except in case of a small ore-body lying within two adjoining 40's.

The outline of the ore-body is established from the drill holes. Beyond the outer or limiting drill holes there is some ore which is estimated as follows: On each cross-section the ore is followed beyond the last hole in ore to the point where the section indicates a thickness of ore equal to half the thickness shown in the drill hole. For estimating purposes this point is considered the limit of the ore-body. There are two such points to each section. By connecting these points from all the sections the area of the enclosed layer of ore is found.

The process of making a total tonnage estimate is illustrated on Plate I, Fig. 6 is a typical exploration sheet containing the "exploration data" of the Judson Mine, a simple ore-body, devoid of complications. This sheet contains all the data necessary for the construction of cross-sections and at least one longitudinal section. Fig. 7, marked cross-section 12, shows such a section and shows how the various layers are connected. In complicated ore-bodies this work calls for thorough familiarity with the formation and good judgment on the part of the engineer. Fig. 8 shows the outline of the ore-body determined from study of the cross-sections. Fig. 9 presents the ore estimate in detail.

Total tonnage estimates when made by experienced engineers on sufficient exploration data are usually surprisingly accurate. As properties are developed and mined these estimates are revised annually; the year's ore shipments are deducted and the estimate is brought up to date.

Pillar estimates.—The method of making a total tonnage estimate is the same for open-pit and underground mines. In the latter, when the property, or certain layers of the ore-body, are nearly worked out, a "pillar estimate" is made of the remaining ore. The areas of the pillars on each level are measured with a planimeter from maps that indicate the rock boundaries. The sum of the areas of the pillars on each level multiplied by the difference in elevation between the level in question and the one above it, will give the volume of ore in the pillars to that level. Each level and sub-level is successively treated in this way. To the sum of the volumes of all the pillars there must be added the volume of ore remaining below the bottom level of the mine. This is obtained as follows:

Cross-sections are made from the exploration drill holes and from the winzes and underground drill holes sunk at various points from the bottom level to determine whether or not lower levels are necessary. From these the average depth of ore below the level may be calculated. The total area of the bottom level multiplied by this average depth will give the volume of ore below it. Each mine determines a cubic foot per ton of ore factor by measuring openings in ore for a certain time and keeping a record of shipping weights during the period. By dividing the total volume of ore by the factor thus obtained the remaining tonnage is determined. The McQuinn Mine (see Fig. 10, Plate I) was selected as a sample illustration of a pillar estimate. The following is a statement of the computations that enter into this estimate:

No. of level	Area in sq. in.	Height of each	Area drift and incline sq. ft.
Bottom level	16.21	18 feet	....
416 feet	14.82	12 feet	9282
428 feet	16.86	12 feet	115
440 feet	14.10	12 feet	3027
452 feet	6.93	9 feet	3956

Bottom level, 2500 × 18.00 × 16.21 equals 729,741 cu. ft.
416 feet, 2500 × 12.00 × 14.82 equals 444,600 cu. ft.
428 feet, 2500 × 12.00 × 16.86 equals 507,800 cu. ft.
440 feet, 2500 × 12.00 × 14.10 equals 423,000 cu. ft.
452 feet, 2500 × 9.00 × 6.93 equals 155,925 cu. ft.

Total gross volume............2,261,06

Volume in drifts and incline to be deducted:
Bottom level.
416 feet.............12.00 × 9282 equals 111,384 cu. ft.
428 feet.............12.00 × 115 equals 1,380 cu. ft.
440 feet.............12.00 × 3027 equals 36,324 cu. ft.
452 feet............. 9.00 × 3956 equals 35,604 cu. ft.

Deduct for drift and incline.... 184,69

Net cubic feet of ore..........2,076,37

This net volume of 2,076,374 cubic feet, with a factor of 15 cubic feet per ton gives 138,423 tons. Samples taken in the drifts and raises established an averag analysis of iron 58.77 per cent, phosphorus .031 per cent for this tonnage. The ma submitted is a reduction from a scale of 50 feet = 1 inch.

WESTERN MISSABE ESTIMATES

The ore formation of the Western Missabe is described on page 177. The soft ness of the strata practically limits drill prospecting to churn drilling. Since muc of the ore contains a varying proportion of sharp fine sand (25 per cent to 75 pe cent, with 35 per cent to 40 per cent in the normal ore), it is difficult, if not impossibl to judge from the drillings, which consist wholly of sharp fragments, whether th stratum passed through is taconite or paint-rock, or whether it is ore-material tha can be commercially washed.

Recent development by test pits and raises has disclosed large quantities o "paint-rock." One of the distinguishing features of paint-rock on the Eastern Missab is the presence of comparatively large percentages of alumina and phosphorus whic are readily determined by analysis. On the Western Missabe the paint-rock contain so little alumina and phosphorus that the chemical analysis of drill samples is insuf ficient evidence upon which to distinguish between paint-rock and ore. Until recentl ore estimates in the Western Missabe district were based wholly upon chemical analy sis. Three grades were established: First, a non-Bessemer shipping ore containin

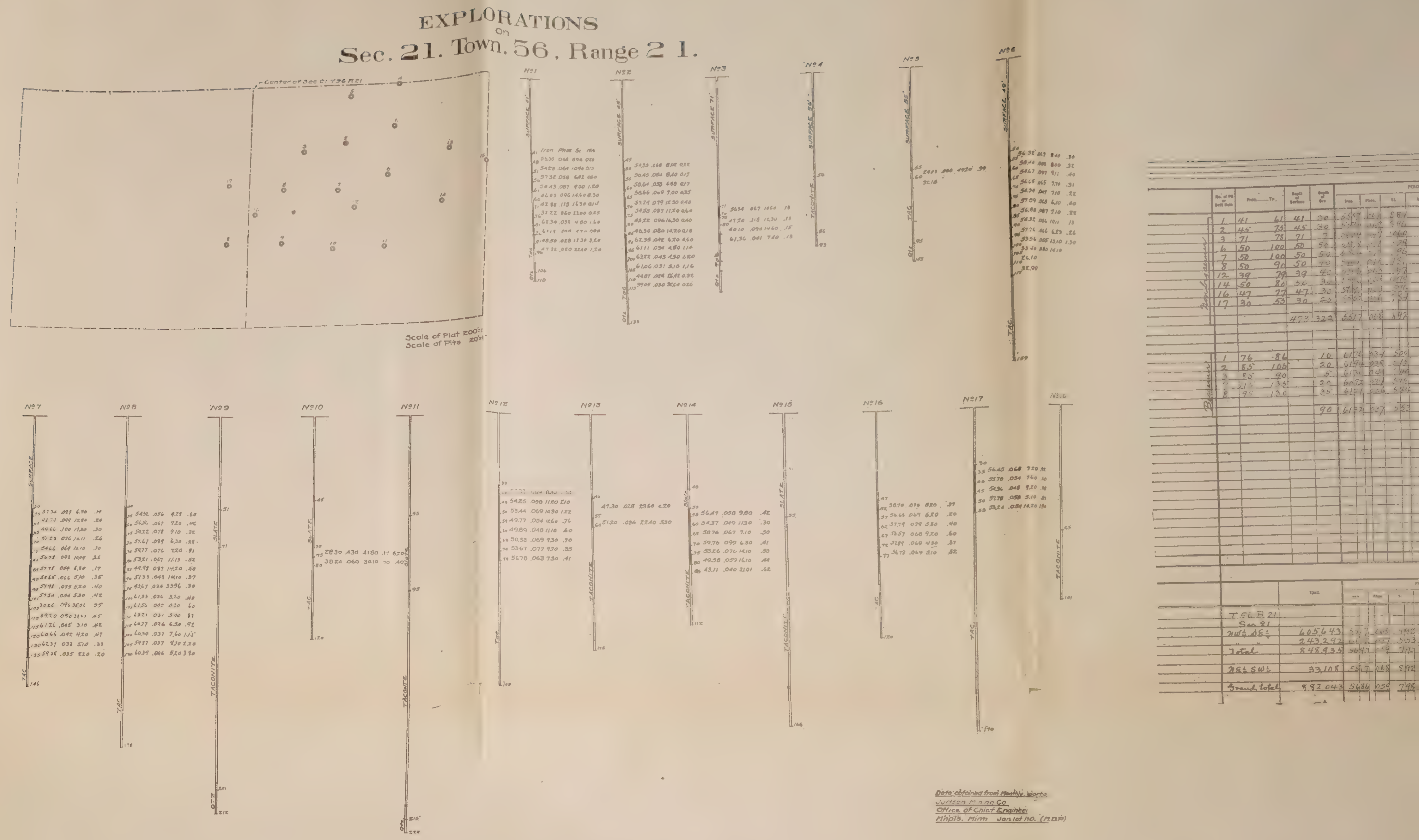


FIG. 6—Exploration Sheet

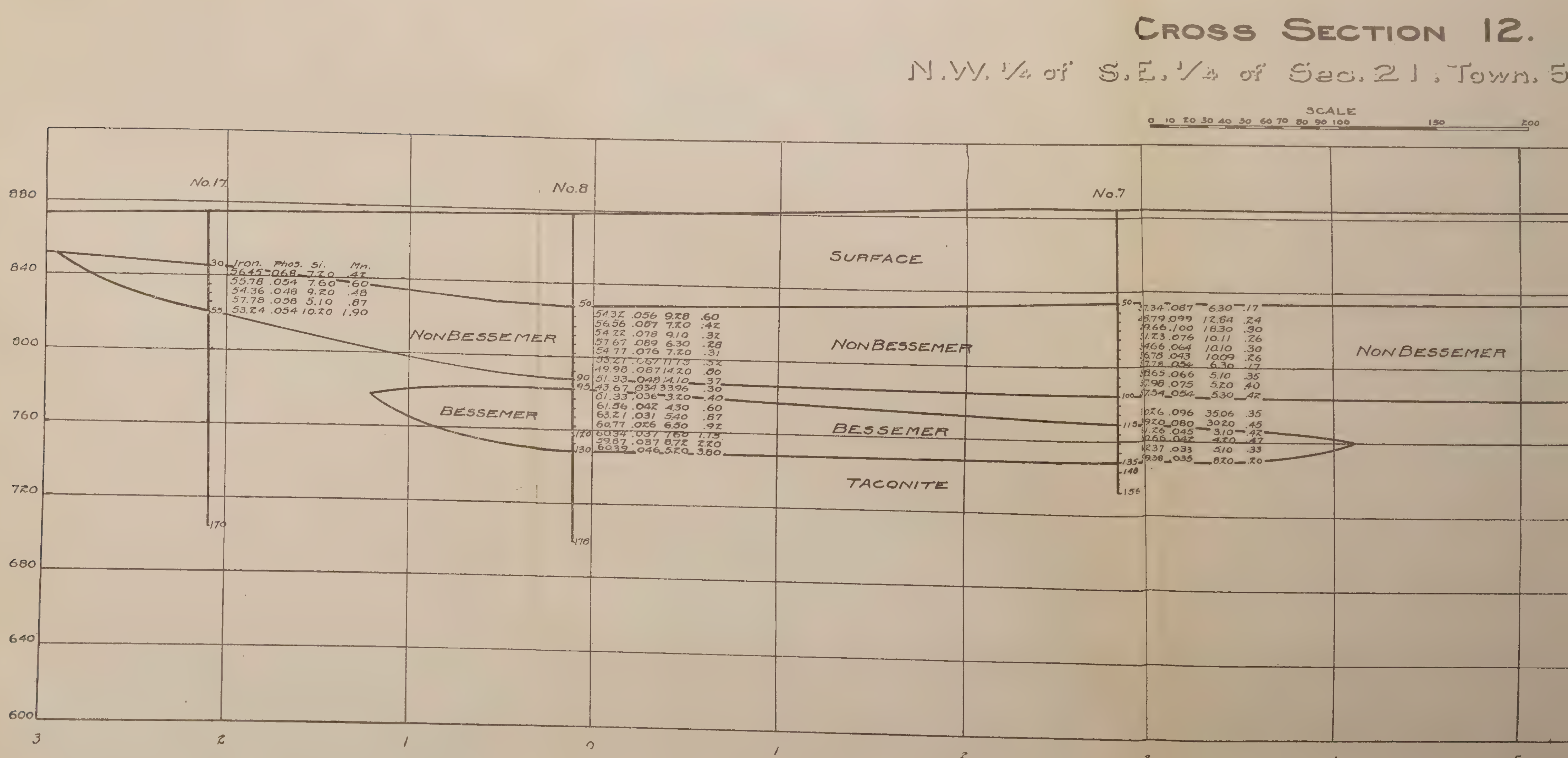


FIG. 7—Cross Section

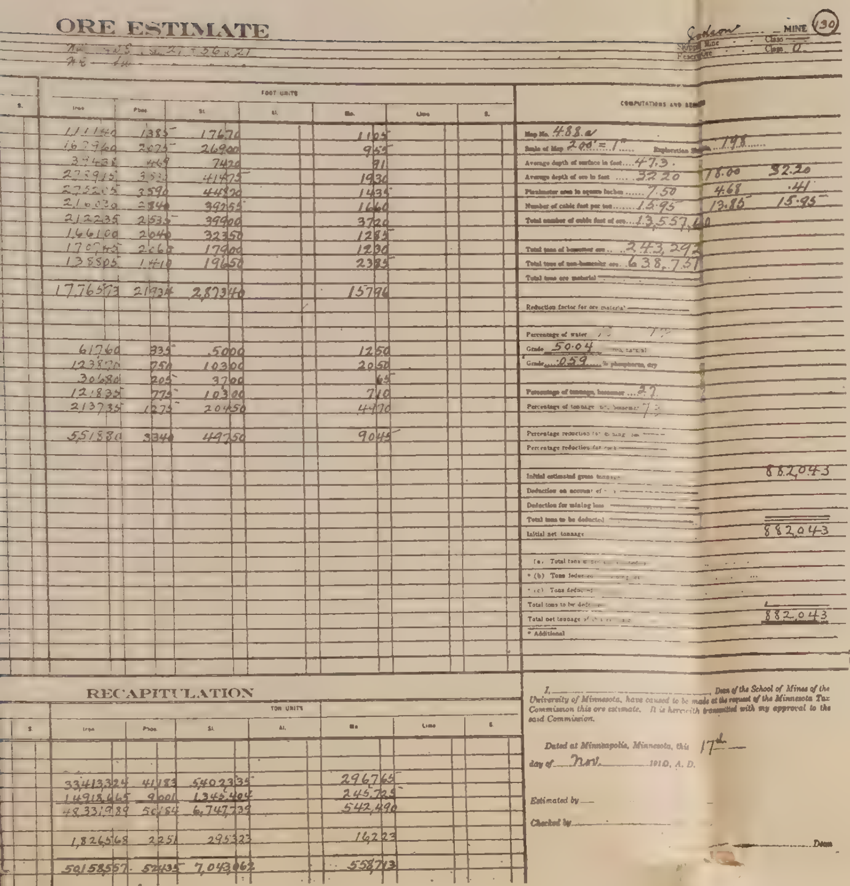
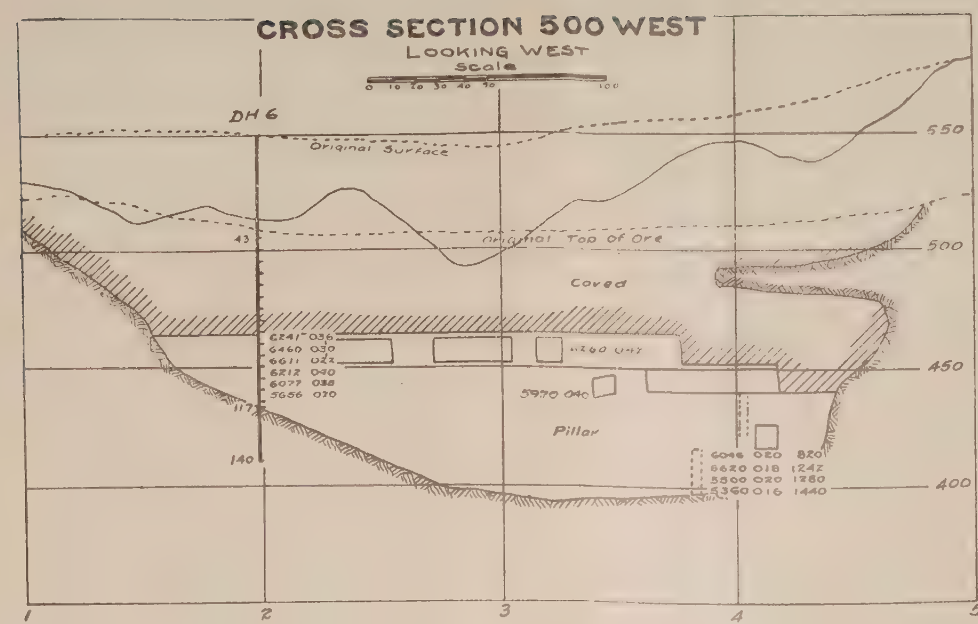


FIG. 9—Detailed Ore Estimate



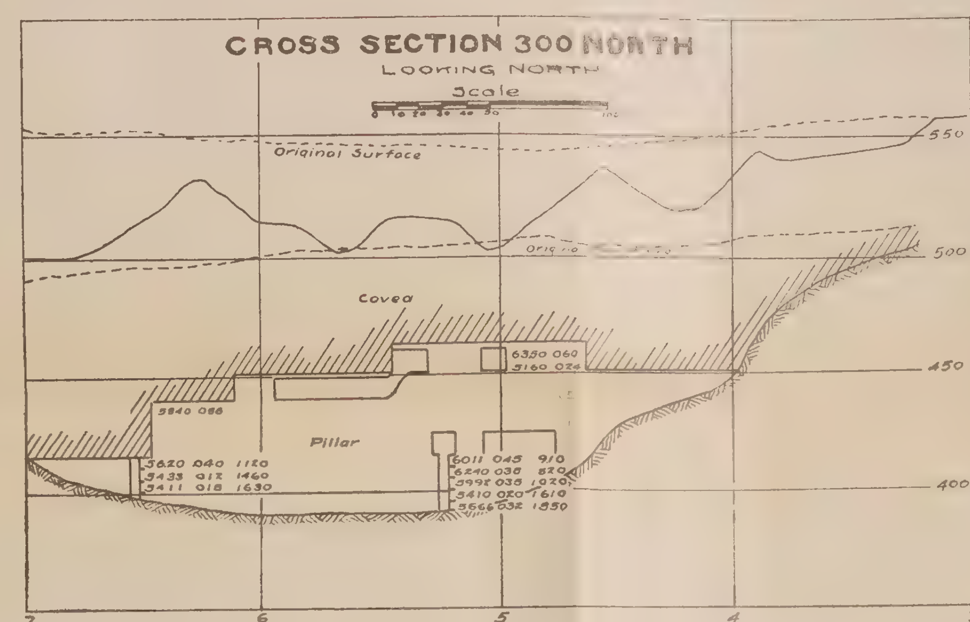
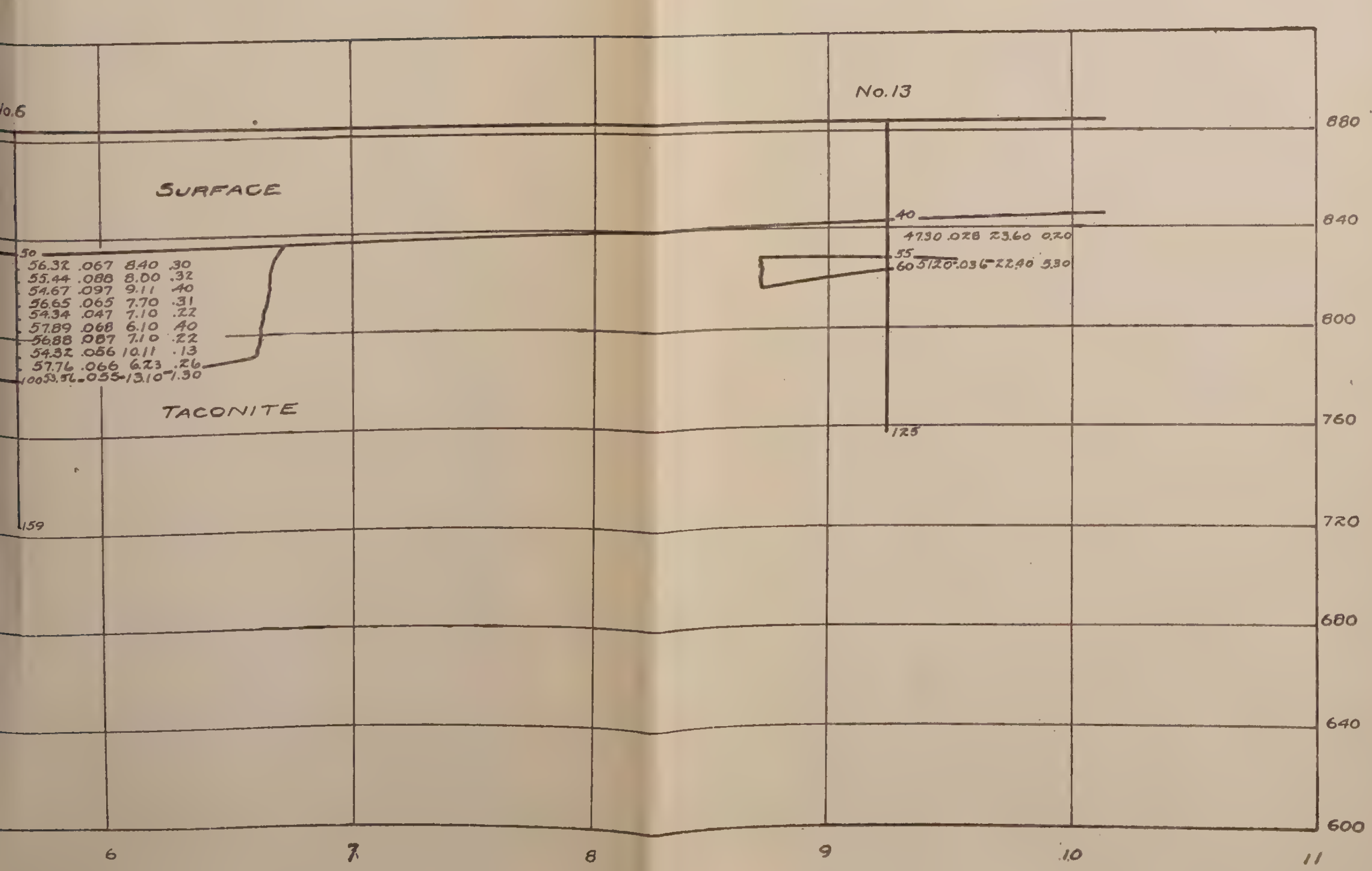
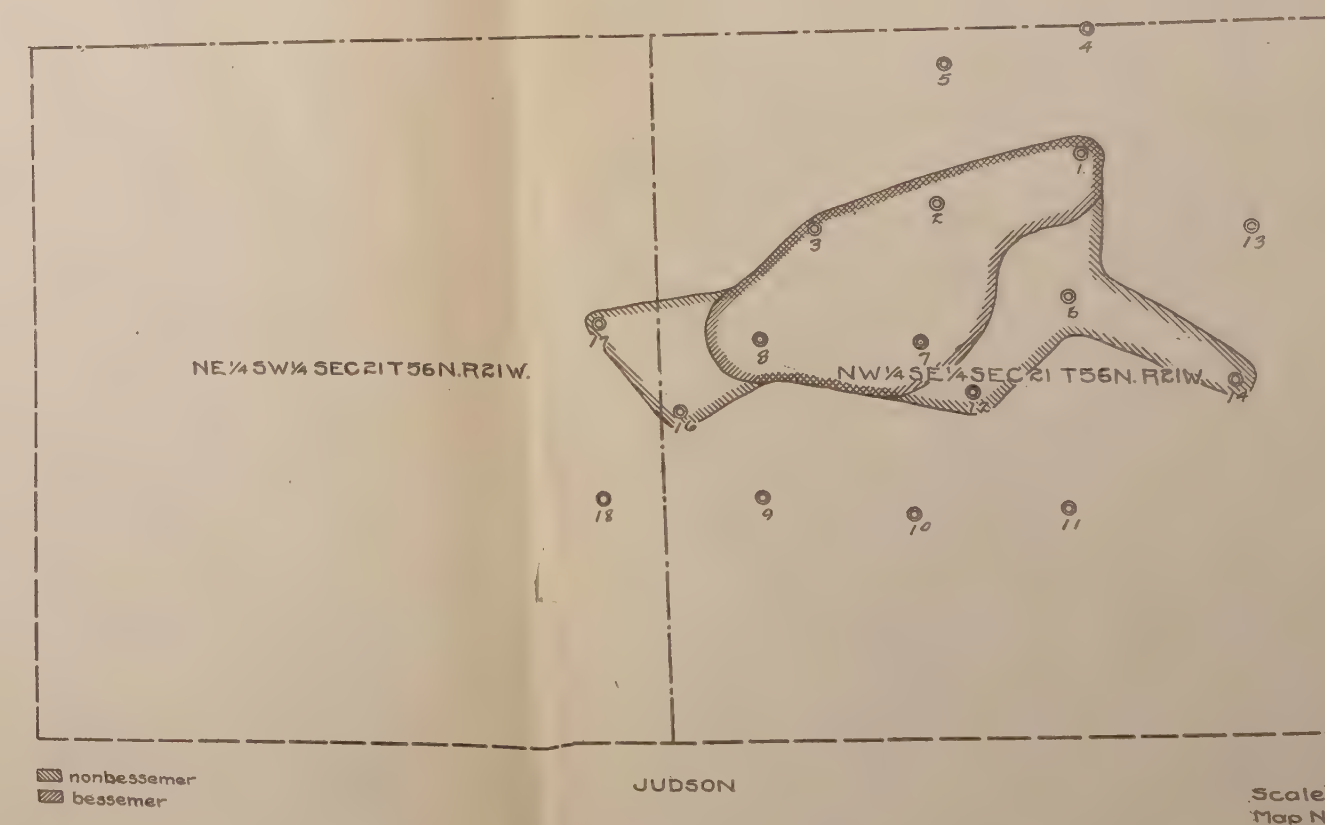


FIG. 10—Pillar Estimate

NOTE The 428 ft Sub Level is worked back even with the 416 ft Level and is therefore not shown.

Range 21.





Scale
Map No.
Jan 1st

FIG. —Outline of Ore-body

57 per cent iron and 0.04 per cent phosphorus of which quite a little tonnage was developed; second, a first-class silicious ore averaging 49 per cent iron and 0.04 per cent phosphorus; third, a second-class silicious ore averaging 40 per cent iron and 0.04 per cent phosphorus. It was assumed that the bulk of the silicious ore could be successfully raised to merchantable grade by washing. The sinking of test pits quickly showed a wide variation in chemical analysis of churn drill samples and test pit samples from the same stratum. This variation was caused by a washing away of silica and alumina and concentration of iron, due to the churning action in the drill hole. The alumina in some cases was reduced from 17 per cent to 0.2 per cent while the iron was correspondingly increased, in some instances sufficiently to make drill samples run 7 per cent higher than test pit samples.

This selective or concentrating action within the drill hole might readily make a poor class of paint-rock appear to be a good grade of non-Bessemer ore; it might readily show a large tonnage of washable ore in ground which really averaged as low as 30 per cent iron. A number of comparisons between drill hole and test-pit data show that a concentration of iron in the drill hole is inevitable. The degree of concentration varies greatly, but would seem to average about 4 per cent. Present-day estimates in the Western Missabe district are made with success only after the ore body has been thoroughly tested to determine the ratio of washable to non-washable ore and the ratio of concentration possible.

The technical man realizes that estimating is wholly a question of applying certain principles to the particular case in hand. It requires thorough familiarity with local conditions, practical experience in the work, and nice judgment. A more extended discussion of local Missabe practice would involve too much detail to be of interest to the average reader.

MINING METHODS

To-day the Missabe Range is famous among mining men as a locality where engineering talent, capital, and business organization have solved many hard problems—a locality that sets the standard for open-pit mining and for that variety of the caving system commonly known as "Top Slicing and Caving." Since the present standard methods represent many years of trial and evolution, a brief description of the older methods, now discarded, may be of interest.

Historical.—Mining was begun on the Missabe with pick and shovel, some thousands of tons being mined in 1892 through a shaft on the Iron Mountain Property, now one of the largest of the old open-pit mines with over 17,000,000 tons to its credit. The same year the first mechanical experiment was made at the Biwabik. A small steam shovel was hauled in by wagon and stripping operations were begun. The following year a contracting firm brought in a small, 27-ton shovel, set it on rails and successfully stripped several hundred thousand yards of overburden from the same property. It would be difficult to put into words just what the successful demonstration of the availability of the steam shovel for stripping and mining meant to Minnesota, to the iron industry, and, in fact, to the present generation. The American supremacy in the iron and steel industry is largely the result of the quick development and remarkable output of the Missabe Range. One of the principal factors in the attainment of this position is the adaptation of the steam shovel to mining. Without the steam shovel or some equally flexible, efficient substitute not only would the Missabe Range still be merely in the development stage, but a large percentage of the ore now profitably mined or carried as merchantable reserve would never reach the market.

The Fayal Mine at Eveleth is one of the oldest on the Range. Most of the methods in use during the early stages of Missabe mining were tried out at this property at one time or another. The application of the steam shovel to many ore-bodies formerly opened up and worked as underground mines has exposed the wastefulness of these methods. Much ore was lost on account of the selective mining practiced. Much of the discarded low grade ore became mixed with the sand, becoming a total loss. A large tonnage was left in the form of pillars that have since crushed and caved. Some of the pillar ore has become so mixed with the sand in the rooms as to be a total loss. The extension of the open-pit work has rendered much of this ore accessible. Fig. 71 on Plate VII shows a steam shovel at work on an old pillar after removal of the caved overburden. In the strictly underground sections the maps indicate the position of the pillars in a general way. The expense of

reaching the pillars through crushed, caved ground is heavy and the recovery percentage is often small, making the cost of ore abnormally high.

The area of the Fayal ore-body is over 200 acres. The total shipments, covering 15 years of continuous operation to January 1, 1911, amounted to 19,000,000 tons. The yearly output often reached 2,000,000 tons. The 1910 output was 1,485,099 tons, about 30 per cent of this being underground ore. The 1911 output was 434,364 tons, all underground ore. About two-thirds of the ore-body is mined out. Prospecting was begun in 1894. A dozen test pits were sunk on a 40-acre tract to a depth of 60 feet. Eight of these went through 25 to 40 feet of drift into hematite ore. A couple of them bottomed in taconite. The balance stopped in ore for want of funds. About half a million tons of 62½ per cent ore was shown up and the lease to the forty was sold for $100,000. The new owners put down an 80-foot, two-compartment incline and ran a drift out into the ore at a depth of 30 to 40 feet below the sand. Meanwhile drill work from the bottom of the test pits showed the thickness of the ore to be 100 feet and over in many places. Adjoining forties were leased and more or less explored. A new shaft was sunk about 1,000 feet away on an adjoining forty. A main tramming level was started 50 feet below the main level at No. 1 Shaft, and a couple of timber slides were raised up to the surface. No. 1 Shaft was deepened and a second main level run to correspond to the main level from No. 2 Shaft.

The main level territory had been cut up into irregular pillars by a number of drifts run quite high and wide, in fact they might more properly be called rooms or stopes. There was no thought of economy or system. Ore was simply taken out haphazard. Considerations of safety at last compelled the timbering of some of the larger openings and a Room and Pillar Method with square-set timbering in the rooms was first developed. Raises were run up to within 20 feet of the ore and a "sub-level" was developed from which square set rooms were run, 3 sets (24 feet) wide and several sets high with 24-foot pillars intervening. Sub-levels were started on various horizons below the first one and the square set rooms became larger, up to 7 and 8 sets high, 5 sets wide and several hundred feet long. The rooms were filled with glacial overburden run into them through raises put up to the surface for that purpose.

Rooms began to cave frequently with little warning. The larger rooms required much expensive repairing and bracing. When a large room caved, surface water would often flood the level and block the drift with mud. Instead of cleaning out the drifts new ones were run and the ore was cut up still further. The ore was cheap and not in great demand. Scant attention was given to the conservation of resources. It was simply a question of getting the richest ore out at the least cost per ton mined. The next generation might take out the pillars.

Another large ore-body was soon discovered lying some 1,400 feet in another direction from No. 1 Shaft. The overburden was found to be only 15 to 30 feet deep. It was decided to strip, and a contract for 500,000 yards was let at 35 cents a yard. This ore-body proved to be of such extent that it is still furnishing steam-shovel ore. One portion of the ore-body lay so deep that it was impracticable to haul all of the ore out with locomotives. A main level was developed under the pit and connected with a

150-foot inclined shaft on the edge of the pit. Raises were put up to the pit bottom through which the ore was "milled." In some cases the steam shovel was used, dumping its load directly into the raises.

Open-pit mining produced ore so cheaply and gave such a high extraction that attention was drawn to the unsatisfactory state of affairs underground. It was thought that the milling system, so successful in the open pit, might be adapted to underground conditions. In the ground tributary to the first shaft raises were run from the bottom level to the sand. A wide drift was then run under the sand and timbered as shown in Fig. 11, Plate II, with saddleback stulls and heavy lagging. The rooms were 23 feet wide and up to 100 feet in length. The ore was milled into raises, the walls of the rooms being vertical. When a room was mined out to the bottom, it was filled. From the standpoint of the rooms the experiment was a success. The rooms were mined out with little difficulty at half the cost of square set rooms. They were, however, imperfectly filled and the pillars caved badly. The losses incidental to square set rooms and the pillar losses in connection with saddleback rooms led to the final abandonment of both methods in favor of the top slice method which is now generally used on the Missabe, though it was at first only used as a means of mining pillars that could not be taken out by any other method.

The Chisholm and Hibbing districts developed some modifications of the square-set method—interesting merely from an historical standpoint. One of these is illustrated in Fig. 12, Plate II. Untimbered drifts were run on the main level at 50-foot centers to the property line. They were then timbered back for 5 or 6 sets with square-set timbers. (See drift No. 1.) The drift was next widened out to 3 sets in width and a room mined out 3 sets wide to the sand overhead. Timbers were delivered to the rooms through timber drifts run 40 feet above the main level. These rooms were connected by small cross-drifts at the second set from property line. When the rooms were mined out 3 sets wide, 6 sets long and to full height of the ore, a start was made on the pillar. (See drifts 1 to 4; pillar between 4 and 5 being attacked.) Pillars were worked in vertical stopes carrying a single set from floor to sand above, continuing this until the tier was removed across the pillar. The entire pillar was thus removed flush with the adjoining room work. Fig. 12 shows two tiers removed from the pillar between 4 and 5 and the third tier started; the pillar between 6 and 7 lacks one tier of complete removal. The rooms were mined full width on and above the second floor. The pillars were mined in vertical runs one set wide. When the rooms finally caved, a pillar of ore 3 to 4 sets wide was left parallel to the property line and the operation of opening rooms repeated. These pillars were not square-set; they were split by new drifts from which raises were run up. The pillar was then sliced out by the slicing method to be fully described.

Another method is illustrated in plan and section in Fig. 13, Plate II. As in the former method untimbered drifts were run and were then timbered back 5 to 6 sets and finally enlarged to 3-set width. The caps of these sets were carefully lagged. The one was then shot down for a height of 5 feet above the timbers, and the loose dirt left in the lagging to serve as a cushion. The sub-level was run 40 feet over-

head. From the end of the room (at property line) a raise was run up to cut the sub-level and reach the sand overhead. The sets on each side of the drift-set were provided with chutes, the caps were propped to take the weight of the blast, and the second round of holes blasted. This blast would break the ore for about two sets along the drift. Thereafter groups of 8 or more holes were put in radically from the sub-drift at 10-foot intervals. The three downward holes were charged with black powder and the others with dynamite. As many as 500 cars holding 42 cubic feet have been loaded from a single round. These stopes were sometimes drawn back over 100 feet before they caved. The pillars between the rooms or stopes were split by drifts and 4 to 5 subs prior to slicing.

MODERN MINING METHODS

Missabe ore-bodies may be roughly divided into two classes—those requiring underground methods, and those susceptible of being stripped and mined with steam shovels. During the last few years open-pit ore has materially exceeded underground ore. This condition will probably continue for some years to come, though in decreasing proportion. It is conceded that fully one-half of the known resources must be mined by underground methods.

PRELIMINARY COMPARISON OF METHODS

So many variable factors enter into the consideration of the relative advantages of open-pit and underground mining that it seems best to confine the discussion of this question to the following statements: The advantages in favor of open-pit mining are so numerous that when the cost per ton of open-pit ore equals or even exceeds by 10 cents the cost of underground ore, a large operator with plenty of capital and equipment would be likely to choose the open-pit method for the following reasons:

1. When a property is once stripped and equipped a much larger daily tonnage can be mined than is possible from an underground mine. There are two reasons for this. The transportation avenues in an underground mine are limited. The amount of ore that can be removed from a given area by the slicing system is limited by a great many factors that do not enter into the open-pit problem.

2. The tonnage per man is many times greater in open-pit work; therefore a given tonnage can be mined with a much smaller crew.

3. The proportion of tradesmen and experienced men is less; there is a larger proportion of laboring men.

4. Operations may be suspended at a moment's notice without incurring heavy maintenance and up-keep charges during the close-down. Resumption of operations does not call for special expenses and preparations.

5. Underground mines are planned for a specified daily tonnage. Circumstances often require a large reduction and sometimes a large increase in this tonnage. In either case the economy of operations is materially affected and there results a decided increase in the cost of ore.

The principal objections to be recorded against open-pit mining are:

1. The fact that in most cases it calls for a longer development period and a far greater initial outlay of capital than is required for the development and equipment of the average underground mine.

2. The difficulty in making grades. It is often stated that open-pit mining has an advantage over underground mining in this respect. Such is not the case, especially in spotted ore-bodies. In underground work the unit of operations is a party of two miners whose daily output may run up to 30 tons under very favorable conditions and there are from 25 to 40 or more working places to draw from. In open-pit mining the unit is a steam shovel with a capacity of several thousand tons and in a spotted ore-body it is very difficult to maintain desired grades without seriously affecting the economy of mining.

Operating estimates.—When a property has been drilled and estimated, the engineer makes further estimates to determine the method of mining best suited to the ore-body under consideration. The following basis has been established for comparison of underground and open-pit mining costs:

TABLE NO. 1

Stripping ordinary glacial drift, 30 cents a cubic yard.
Stripping ordinary paint-rock, 30 cents a cubic yard.
Stripping ordinary broken taconite, 75 cents a cubic yard.
Stripping ordinary solid taconite, $1.00 a cubic yard.
Steam shovel mining, ordinary ground, 15 cents a ton.
Underground mining, ordinary conditions, 75 cents a ton.
One cubic yard of ore is roughly 2 tons.

Sometimes a glance at the ore estimate will suffice to classify part or all of an ore-body. Often a calculation must be made as exemplified by this case: A drill hole shows 50 feet of ordinary glacial drift and paint-rock, 15 feet hard taconite, 36 feet merchantable ore. All other things being equal, is this an open-pit or an underground proposition?

Reducing the consideration to a column of one yard square at the drill hole, a comparison may be made using the data in Table No. 1.

Underground mining—

Cost of mining a column of ore one yd. sq. and 36 ft. high @ 75 cents a ton (1 cu. yd. = 2 tons), $36/3 \times 2 \times \$0.75 =$ $18.00

Open-pit mining—

Stripping a column one yd. sq. and 50 ft. high of glacial drift @ 30 cents a yd., $50/3 \times \$0.30 =$ $5.00

Stripping 15 ft. of solid taconite @ $1.00, $15/3 \times 1 =$ 5.00

Steam shovel mining 36 ft. of ore @ 15 cents a ton, $36/3 \times 2 \times \$0.15 =$ 3.60

Total cost of open-pit work.................................. 13.60

Difference in favor of open-pit..................................$ 4.40

The figures given in Table No. 1 may be reduced to vertical depth for one yard square and stated as follows:

TABLE NO. 2

Cost of stripping glacial drift, 10 cents per ft. of depth.
Cost of stripping ordinary paint-rock, 10 cents per ft. of depth.
Cost of stripping broken taconite, 25 cents per ft. of depth.
Cost of stripping solid taconite, 33⅓ cents per ft. of depth.
Open-pit mining, 10 cents per ft. of depth.
Underground mining, 50 cents per ft. of depth.

Testing the same drill hole according to Table No. 2 the same results are obtained for a column a yard square:

Underground mining—

36 ft. of ore @ 50 cents a ft. vertical depth		$18.00
Open-pit mining—		
50 ft. glacial drift stripping @ 10 cents	$5.00	
15 ft. solid taconite stripping @ 33⅓ cents	5.00	
36 ft. of ore @ 10 cents	3.60	
Total cost open-pit ore		13.60
Difference in favor of open-pit		$ 4.40

This offers a ready method of preliminary comparison to be supplemented by more exact figures when special considerations enter. It is of course understood that such questions as adverse or favorable topography, accessibility of dump room, quicksands, swamps, etc., have a special bearing on each individual case that does not admit of generalization.

The economical limit of stripping is at present considered to be within the following proportions:

1. One yard of overburden to one ton of ore.
2. Not to exceed 2-foot depth of overburden to 1-foot depth of ore. Hard slates and taconite cost from two to three times as much to strip as ordinary glacial drift and it is customary when applying these figures to consider 1 foot of such material as equal to 3 feet of overburden.
3. A maximum stripping depth under any considerations of 150 feet.

PRESENT-DAY UNDERGROUND MINING METHODS

There is to-day one standard underground mining method on the Missabe Range, "top slicing and caving," locally called "slicing." In brief, the working of an ore-body by this method requires the following operations, mentioned in their logical order:

1. Opening a tract of 40 acres or less with a shaft from which a main tramming level is run in the deepest portion of the ore trough. In addition to the hoisting shaft, from one to three timber shafts are sunk at convenient places. If the ore-body is more than 50 to 60 feet thick, it is divided into a series of horizontal layers

each from 30 to 50 feet in thickness, having a main haulage level on the bottom of each layer.

2. Dividing the territory above the main level into slices or sublayers varying from 8 to 16 feet in height, averaging perhaps 10 to 11 feet. There are from 2 to 5 of these layers, each one opening by a sub-level from the manway of the shaft.

3. Cutting up the ore in these sub-layers into a series of long ribs by running parallel "sub-drifts" (6 by 7) at 50-foot centers. The main level is similarly developed and care is taken to make the successive sub-drifts overlie each other.

4. Cutting up the ribs into pillars, 50 by 100 feet, by means of cross-cuts or "extraction-drifts." These are run at 100-foot intervals at right angles to the sub-drifts. The ore is a soft, earthy hematite mixed with some limonite. It is sufficiently firm so that small drifts will usually stand for several years without timber.

5. The ore mined from the subs is trammed to conveniently located chutes terminating in pockets on the main level. A number of raises, 4 feet or more square, are run up for interlevel communication.

6. Slicing is then started at the extremity of the ore-body, or at the property line, by drawing pillars in through the extraction-drifts toward the shaft and caving the roof. These pillars (each 50 by 100 feet) are removed by a series of parallel, contiguous, timbered drifts, 7 feet wide, and carried the full height of the ore, the removal beginning at the top of the pillar farthest from the shaft.

The slicing is usually contract work, the contractors tram their ore to these chutes, whence it is drawn by trammers working company account.

7. The roof in the slices is temporarily supported by light drift-sets. When the ground becomes too heavy, the rest of the ore pillar is boarded up, the floor of the worked-out portion is also boarded or slabbed, and the roof is dropped or "caved" by shooting down the supporting pillars. This boarding up of the exposed ore surface prevents mixing of ore and waste.

The above is a brief outline of the system. The details of its application will now be discussed. For the drawings used in the following description and for some of the detail on pages 46 to 64, I am indebted to L. D. Davenport, of the Oliver Iron Mining Company's Engineering Department, Chisholm District. A 40-acre tract is assumed underlaid by a typical ore-body and the development is planned in accordance with current practice. (See Plate III.)* From the exploration sheets, cross-sections through the drill holes are made, from which top- and bottom-contours are worked out and mapped, and the ore channels located. These contours show the Nels ore-body to occupy a shallow east and west trough with greatest depth at its western end. A main shaft is accordingly located at the west end of the trough and sunk through the deepest part of the ore to the taconite below. In this position it will always drain the ore and, with the exception of the shaft pillar, all the ore can be hoisted through this shaft.

*The cross sections of the Nels ore-body are intentionally omitted from Plate III. The development plans shown serve to outline the general plan of operations and to give an idea of the appearance of a Missabe underground mine in successive stages of development. In order to avoid raising questions as to engineering judgment it seems best to omit the cross sections.

Two principal factors must be considered in locating the main tramming level. It should be as near the bottom as possible so that the greater part of the ore will be mined above it; and furthermore it should encounter the bottom rock at the point from which it is desired to handle the ore from the levels above. If the formation pitches steeply, it may be necessary to use two or even three levels in order to stay near the bottom and yet cover the desired territory. When the deposit is very thick, it is usually divided into a series of horizontal layers, each one with its own tramming level.

The starting point of the main tramming level having been determined, the next step is to cut the shaft station, ore-pocket, pump-room, and sump. Generally the shaft is continued about 40 feet below the main level; this allows 20 feet for the ore-pocket, 10 feet for the skip-pit, and 10 feet for a sump below the latter. It is desirable to place the pump-room and the loading chute of the skip-pocket load at the same elevation. The pump-man can then act as skip-tender unless there is enough work to keep two men busy. The details of the station and the ore-pocket will be found on pages 78, 79. Upon reference to the maps of the Nels Mine (Fig. 14, Plate III) the main level is shown opened as two parallel 10 by 9-foot drifts at 90-foot centers, connected every 150 feet by 8 by 9-foot loop-drifts. A horse of taconite is shown at 1,050 east and at 1,160 east in the north and south drifts respectively. This made it necessary to take up 3 to 4 feet of taconite. The Missabe miner is not a hard-rock miner; he often goes to a great deal of unnecessary expense in order to avoid rock work using inclines and puffers (small hoists) where he might better have driven through several hundred feet of rock at negligible cost per ton of ore mined, though the actual drifting expense in rock might be several thousand dollars.

Drifting is usually contract work, a contract gang consisting of 2 men on each ten-hour shift. Two gangs—8 men in 24 hours—will advance the drifts about 100 feet a month. This includes timbering and putting in "open-sets" for the loops which are later run by a third gang.

When the main level face has advanced 100 feet or so, another gang is started raising. These raises (marked 1, 2, 3, 4, etc., on the plan) have plank chutes and quarter-pan gates. At the proper point in the shaft a fifth gang starts the 177-foot sub level, a 6 by 8 timber drift run due east. Open-sets are put in at 50-foot intervals for cross-cuts. As the sub level gang advances other gangs are started driving these cross-cuts (6 by 7 feet) north and south, intercepting the raises started from the main level. Co-incident with the development of the sub-level, timber-shafts are started through the surface drift to the top of the ore. No. 1 timber-shaft, situate within the first main-level loop, stopped sinking when it reached the ore. A 4 by 4-foot raise is run from the bottom level to connect. After this raise holes through it is trimmed or stoped down to full size and timbered. Timber shafts vary in size from 6 by 6 feet to 6 by 9 feet.

The cross-sections show a horse of taconite extending into the ore-body from the east property line, leaving a narrow channel of ore which runs northeast through exploration drill-holes 316 and 320. A branch drift is accordingly turned off on the

main-level at 500 east. This strikes the bottom rock at 900 east and is so shown on the map. A raise (No. 50) is located directly in line with, and north of, raises 37 and 38. This raise is continued to the capping. A 6 by 7-foot drift is run east from raise 50 at an elevation of 762 feet (determined from cross-sections), and cross-cuts are turned off at 50-foot intervals to correspond to the development already begun on the south. A small drift is driven along the property line to eliminate any possibility of trespass on this sub-level. This is customary when the slice drifts run at right angles to the property line. It is unnecessary when they are parallel.

The timber drift on the 177 sub struck paint-rock and soft slates in the back about 800 east; from there on, to its connection with No. 3 timber-shaft, it is partly in ore and partly in capping. The north sub-drift from No. 18 had surface-drift in the back from the last 15 feet and the south sub-drift from No. 20 was cut off by surface 100 feet from the raise.

The need for another sub-level is evident and drifting is therefore started south from Nos. 20 and 22, 12 feet below the 177-foot sub on the 198 sub, and the development is carried on eastward on this sub. The drift from raise No. 27 on the 177 sub is used as a timber drift. All timber required in the northeast corner of the mine on the 198 sub is run through this drift.

The dip of the capping makes it impossible to carry the development of the 177 sub farther east than the drift north of raise 37. The timber for the workings on the 198 sub on the south side is lowered from the 177 sub down the raises east of No. 20.

STOPING OR SLICING DETAILS

Drift slicing.—When the 6 by 7 slice drifts on the top sub reach the shore line or property line, as the case may be, stoping or slicing is started. The proper point for the commencement of slicing is determined by test raises, usually 3 by 3 feet run up at 50-foot intervals to ascertain the height of the ore above the sub-level. The section in Fig. 18 may be taken to illustrate any drift run north from the sub-level to the shore line. The test raises to the overlying drift are shown. If the ore is found to be less than 11 feet for the last 100 feet of the drift, slicing is begun.

Dropping back from the breast to the point Z where the ore is about 7 feet high a cross-cut is driven 40 feet east and 10 feet west from the center of the sub-drift taking the full height of the ore and boarding up the south end. The sketch shows regular timbering. Frequently the caps are placed parallel to the direction of the cross-cut instead of at right angles. The sets are then called "open sets." These are more convenient since slices may be started from the cross-cut without changing the timbers.

From the end of the 40-foot cross-cut the first slice is run north over the rock until the ore becomes so thin that further advance is impracticable. Often 24- to 30-inch posts are used in the last set. Track is laid in the cross-cut. Usually at the shore line the rock rises so fast that it would not pay to make a turn and extend the track into the first slice. When the rock jumps up several feet, the miners often use wheelbarrows to fill the cars. The slice is sometimes started directly from the drift without driving a preliminary cross-cut. This is not so good. There is a distinc

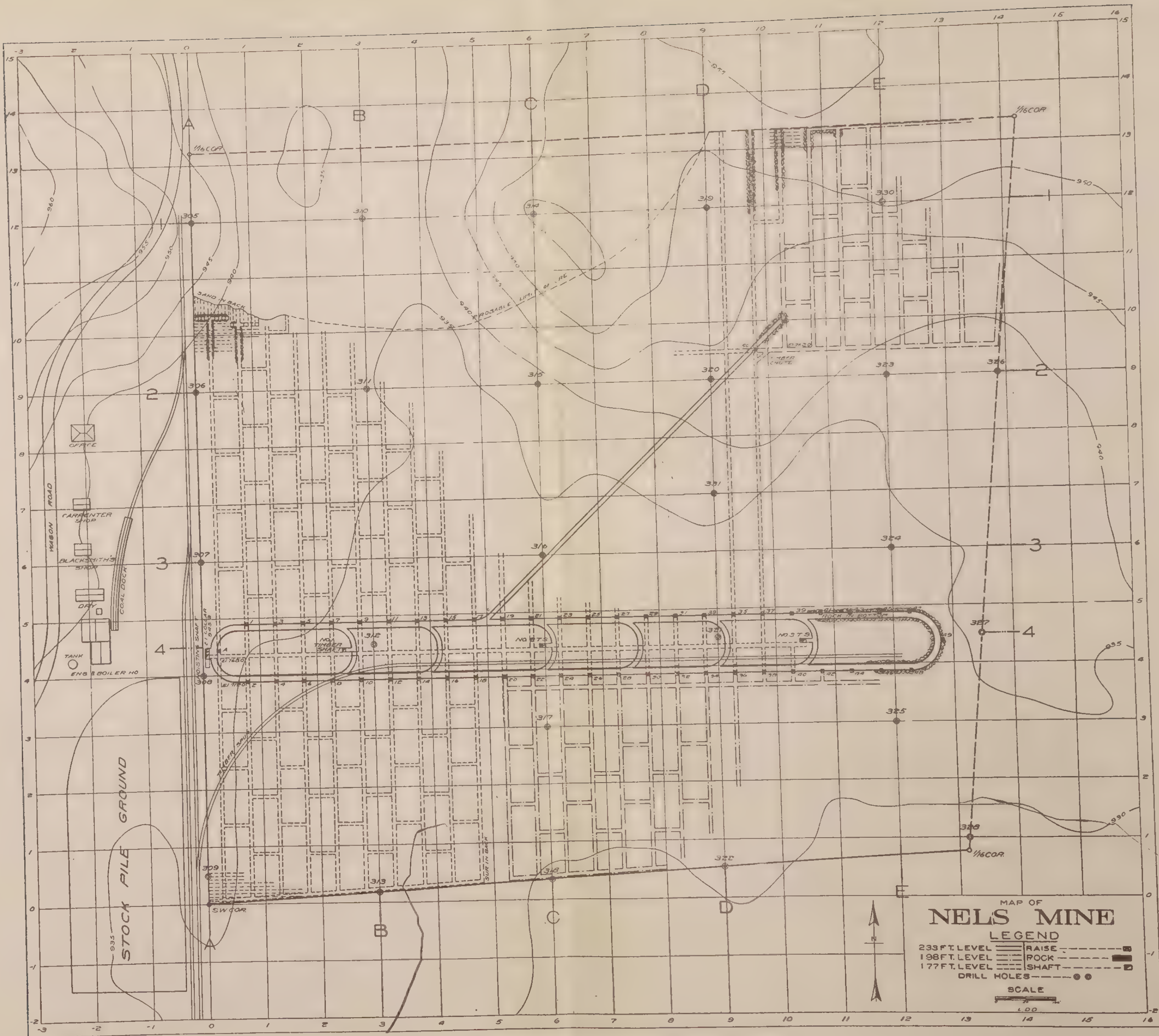
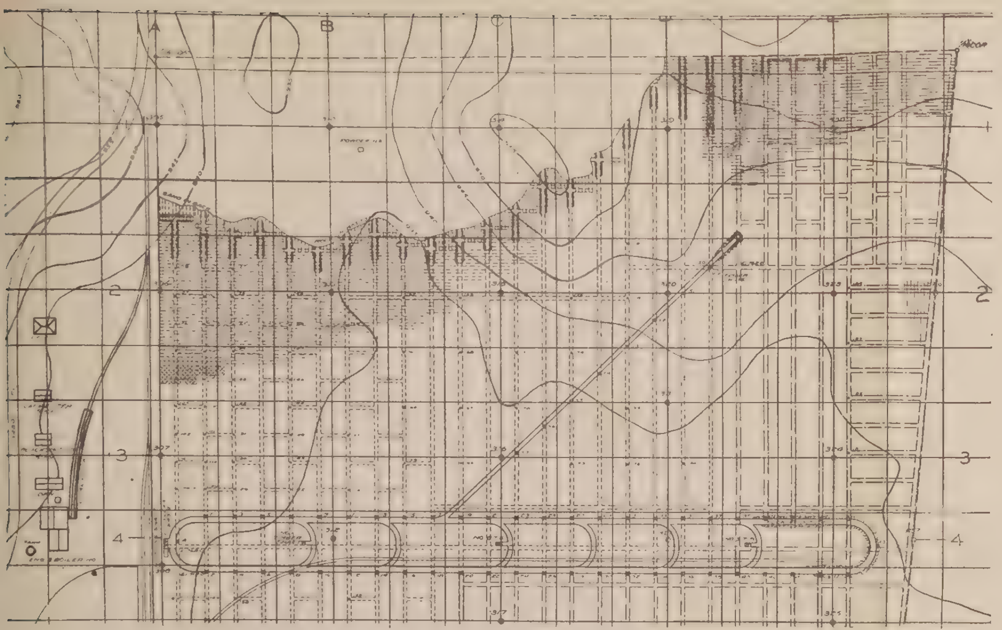
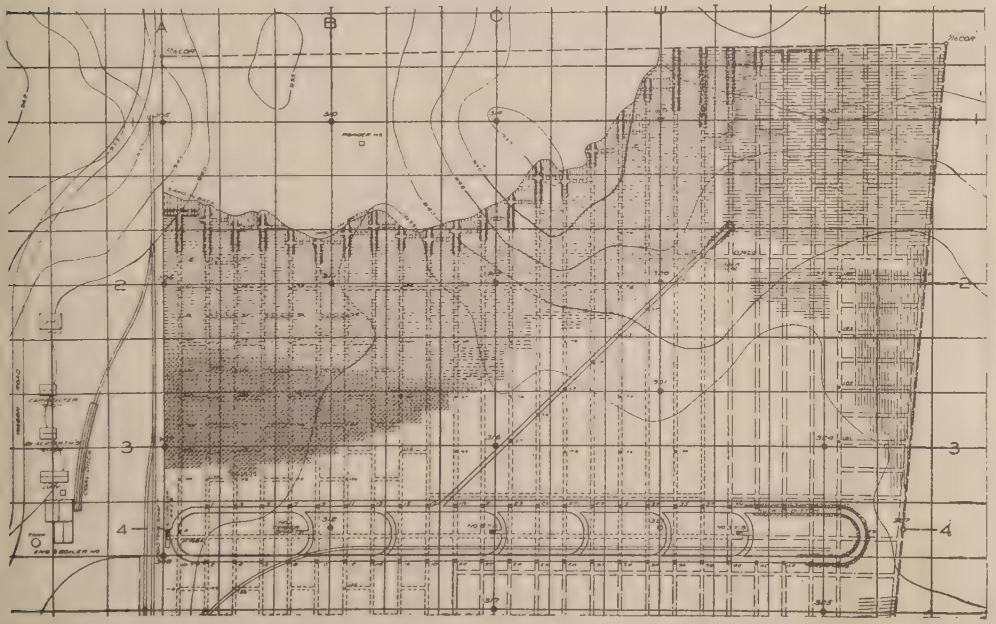


FIG. 14—Development Completed and Slicing Begun





FIGS. 15 and 16—Subsequent Stages

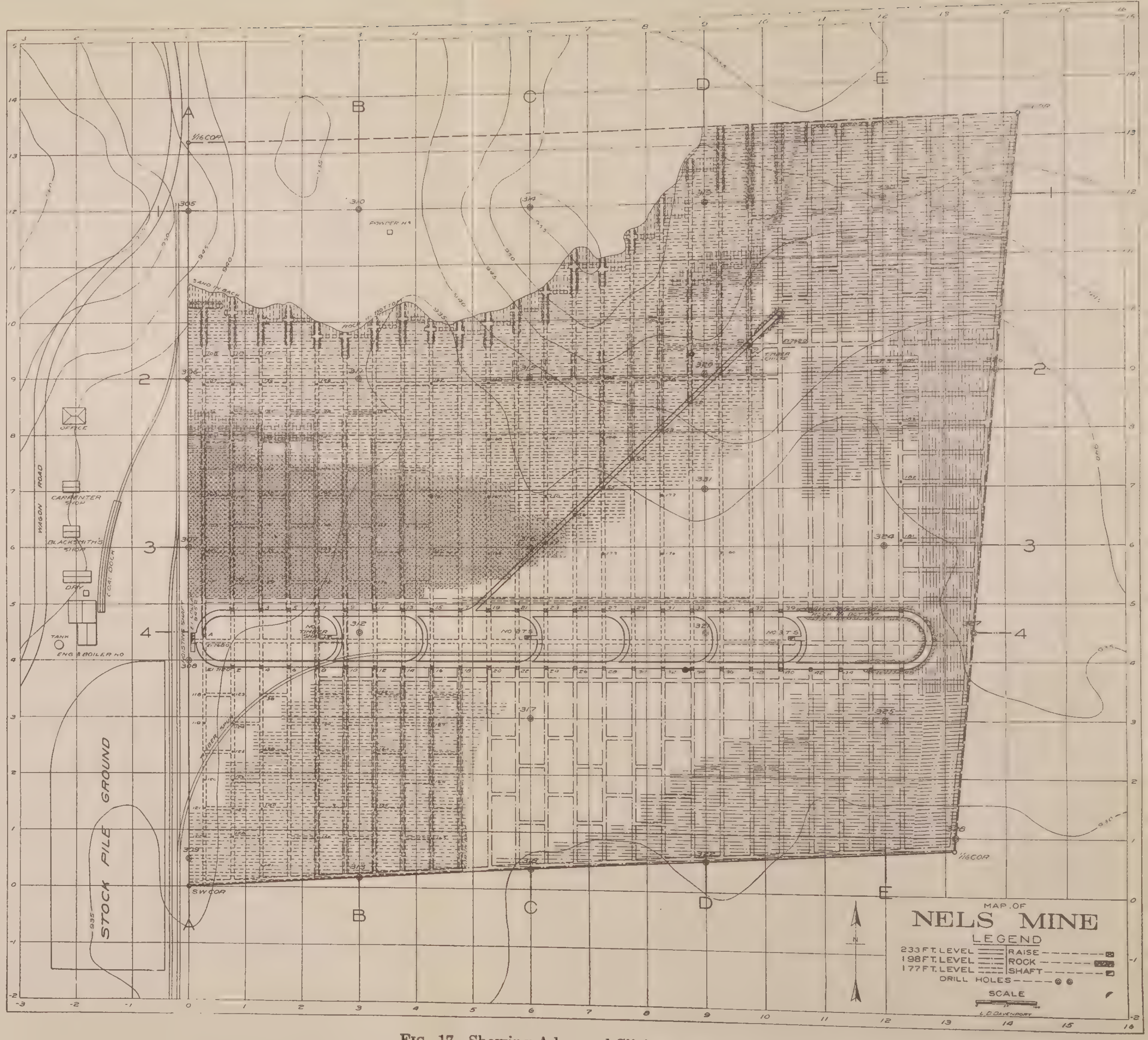
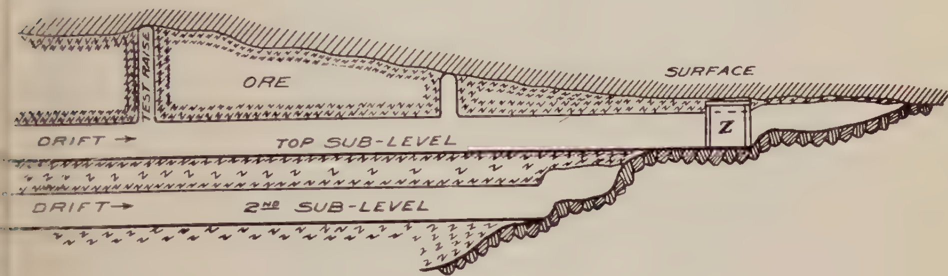


FIG. 17—Showing Advanced Slicing Stage

gain in driving a preliminary cross-cut and slicing up the slope of the bottom rock. The ore comes down grade; the posts of the slice set will have approximately the same length. If the shore line bulges in or out, a slice at right angles can be continued or stopped at will allowing the maximum flexibility. If the ore opens out and the first few slices become heavy, the cross-cut may be breasted, and slices boarded up and the timber blasted down. In this way all of the ground north of the cross-cut Z is mined and caved.



Section—Looking West.

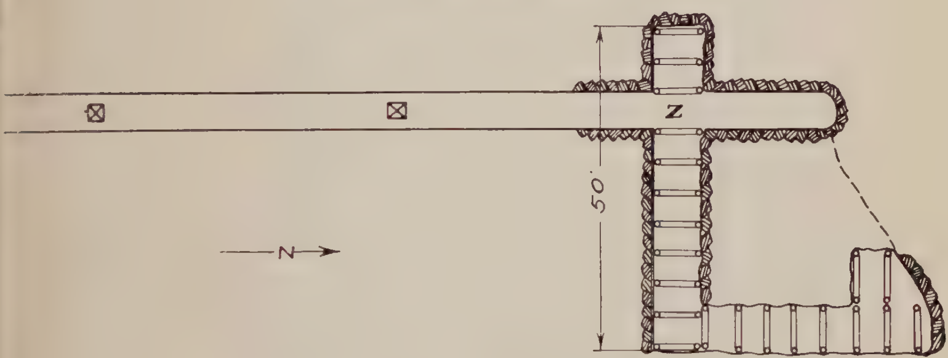


FIG. 18. Plan.

In starting the first room against the caved ground, open-sets are erected in the drift against the caves, with caps parallel to the drifts. The caps and posts are spragged. (See A, Plate IV, Fig. 19.) The slice is continued along the caved ground from A towards B, a distance of 40 feet. The track is turned into the first slice, i. e., nearest the cave.

If the pillar adjoining on the east has been mined and the slice is to strike "end-caves" at 40 feet, it is customary to leave a small pillar of from 1 to 2 feet to hold back the sand. This is indicated on the plan as Pillar X.

The second slice is started as follows: Dropping back to the second or third set in the first slice, say set B on the plan, the first set south of B is mined out, next the set adjoining it on the east, the second set of this slice is mined out to the pillar X. The bottom of this last set is covered with boards and the small pillar X is then taken out, since no harm will be done if a little sand should now run in from the adjoining

caves. The third and fourth sets are then removed, retreating toward the drift, leaving only the pillar *C* which is the last thing to be removed after the four sets on the west of the drift are mined. The solid side of the second slice is boarded up.

Next, the second "open-set" is erected in the drift south of *"A"* set and the four sets (a-a-a-a) on the west side of the drift are taken out and the solid side of the drift is boarded up ready for the cave. Finally the pillar *C* is mined, the room is cleaned out and, if there is ore on the bottom, it is covered with a single layer of No. 5 boards. The drift is then breasted up as shown in Fig. 20, and the room blasted down. The sketch is taken from a point in the drift, showing the outline of the latter in the foreground. The props and cross-pieces hold the boards from bulging in when the room caves. The door or retreat for the miner who completes the charging of the room timbers to be blasted is finally boarded up as shown and the room blasted. Sometimes it is necessary to blast the side of the old caves to make the room fill.

A variation in the detail of slicing where the ore is firm is to begin with two open sets and open two contiguous slices in one operation instead of two. There is then danger that the room may come down before the last two or three sets are mined out and then it is necessary to drive in a new slice and mine the remaining sets from behind. The reason for taking out the pillars with a long and short side, i. e., 40 feet on one side and 10 feet on the other, is that but one curved track is required. The four sets (a-a-a-a) are shoveled directly into the car standing on the drift track.

The end section, Fig. 19, shows the drift with two open sets and the usual method of attacking a slice. Four holes are drilled; holes 1 and 2 are drilled, blasted and squared up. Hitches are also cut in the walls for the posts. This saves staging for the men to stand on. Then (referring to the sketch above) hitches are cut in the top of the breast to hold two or three poles; one end of the pole rests on the cap, the other end projects a foot or two beyond the limit of the squared face and rests in the hitch. Short boards are then placed on top of the poles, which are then wedged tight against the boards. The roof or "back" is now secured and the miners can take up the remaining ground.

The roof usually caves easily and thoroughly after the mine has once been properly started. Frequently a new slice is started in the morning after the night shift has blasted down the old slice; usually it is best to give it a little more time. Fourteen or even sixteen-foot slices are worked with safety, though generally a 10-foot post, giving an 11-foot slice, is preferred and a 12-foot slice is considered the maximum height.

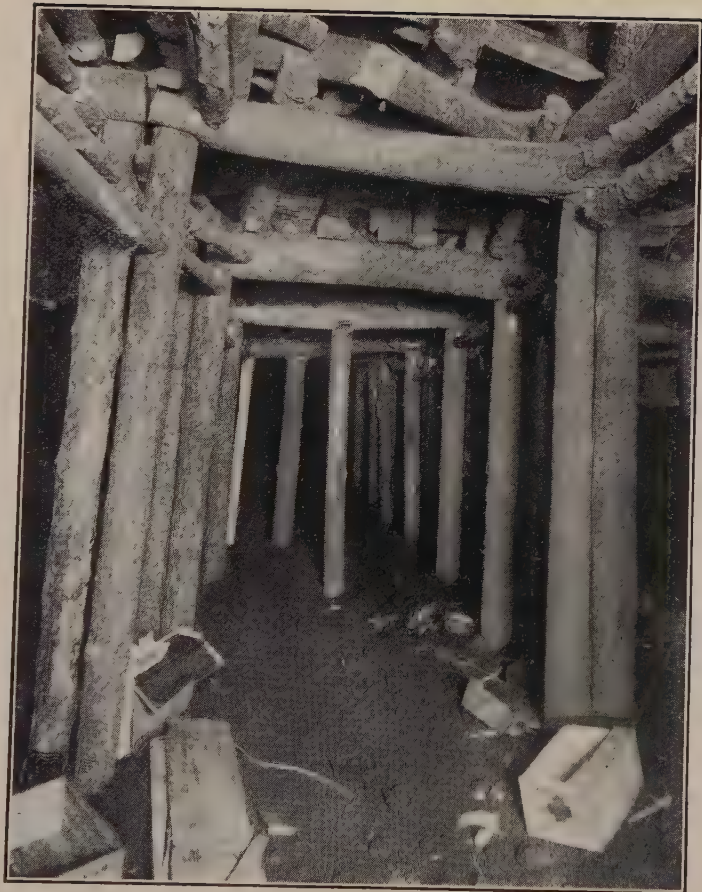
The slicing and caving on the upper of the two subs should always be kept at least 50 feet in advance of the operations on the level below.

Square-set slicing.—The top of the ore-body is usually quite rolling, giving a succession of crests and hollows that frequently have 30 to 40 feet difference in elevation. This condition calls for much irregularity in slicing in the upper layer. Square-set slicing is the simplest and most economical manner of mining these knobs and irregular layers of ore. Square-set slicing is carried on in rooms two or three tiers in width,

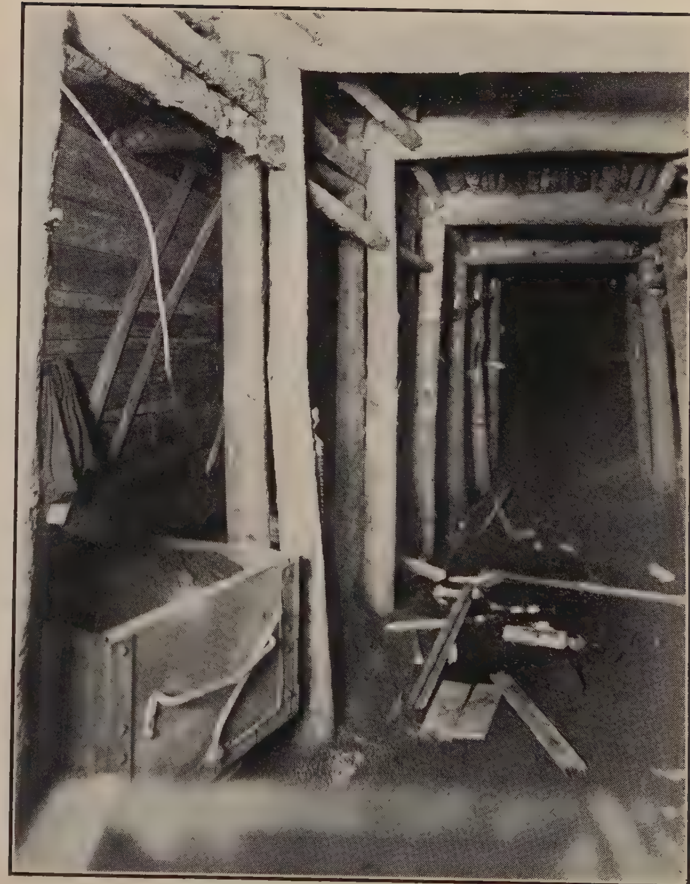
carried from two to four and sometimes five sets high. When such a room is completed, it is boarded up and caved; therefore the process may be properly called square-set slicing to distinguish it from drift-slicing and it must not be confused with square-set mining which has long since been abandoned as unsafe and wasteful.

Fig. 28.—Drift Slice two sets wide, 13 ft. posts. The left breast is taken down in two sections.

Drift-slicing is almost wholly used under an even back or on an even bottom. Square-set slicing is more elastic, permits greater irregularity, and hence can be used to better advantage to mine the tops of the irregular knolls or knobs of ore, to mine under a pitching back or on an uneven floor. Frequently in regular drift-slicing pillars are found that are too high for one slice and would not economically allow two slices.



Typical Drift-Slice.



Drift-Slice boarded up, ready to cave.

Fig. 27

Square-sets can then be substituted. Figs. 21 to 25, Plate IV, show the details of the application of square-set slicing.

The plan in Fig. 21 shows a slice opened by a 6 by 8 untimbered drift to the cave. The first bottom set or set *A* is put in, posts plumb, and caps lined up with the drift. The timbers are set in hitches cut in the walls and tightly wedged. The top of the set is lagged so as to leave an 18-inch opening at the center; this is covered over with short boards laid perpendicular to the lagging, as shown in Fig. 25.

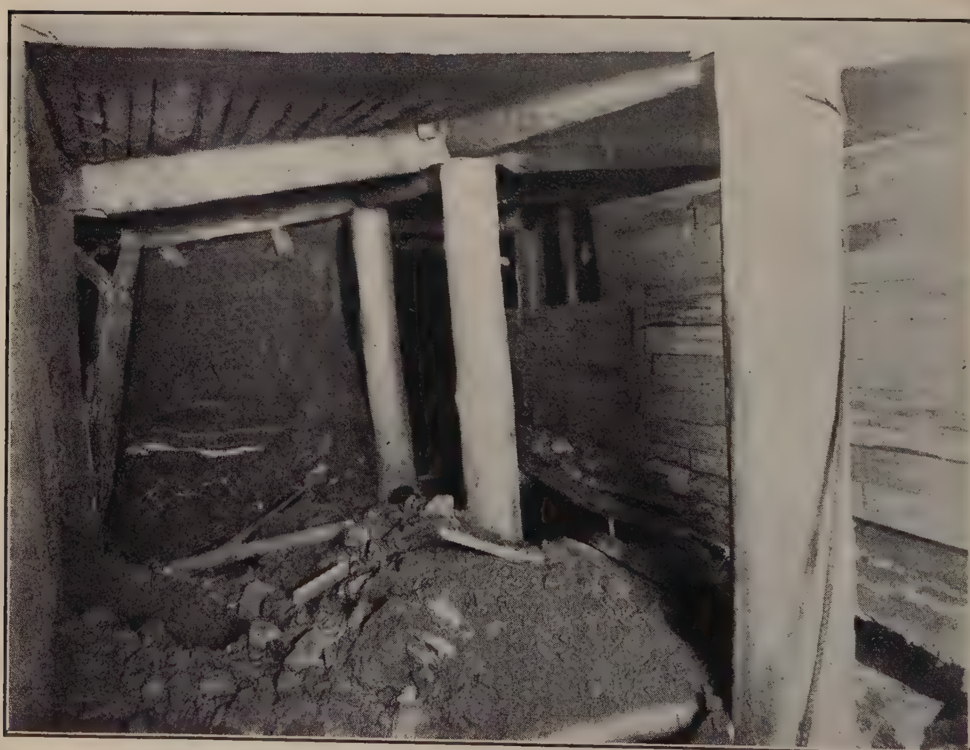
Next *B* set (see elevation) is raised up by cutting in a hole in the back of *A*; before the blast a short rail is propped under the lagging to take the weight of the blast. After the blast, the prop is knocked out, a car is run under the opening, some of the short boards removed, and the ore run into the car. *B* set is squared up and trimmed just enough to allow placing the timbers. *C* set is then raised up without lagging *B* set; the ore from the cutting-in blast drops directly upon the lagging of set *A*. When the last floor (in this case the third floor, set *C*) is reached and the set completed, the back is carefully blocked and wedged solid. Side-lagging is hardly ever necessary. Next *D* set is opened with 3 holes as shown. Lagging and short boards are used on top of *D* as in the case of *A*, only the lagging runs at right angles to the position of the lagging in *A* set. The track is turned so the car can run into *D* under *E* which is blasted and squared up from *B* as *D* was handled from *A*.

The rooms are usually 6 sets (50 feet) long, 5 on one side and one on the other. When a solid tier of rooms is finished to full height, the open set for *A′* is put in and the various operations are repeated with this change, the second set *B′* is blasted from *B* and not from *A′*. The track runs in the first tier and in working out the second tier slab chutes are often used as shown in the sketch. When the room is finished, all the lagging used is dropped from the various floors to the bottom and is stood up in the room as shown in the sketch; the lower end rests against horizontal poles which extend from post to post along the floor. The upper tiers are boarded up with No. 5 boards; props are put in between the posts from cap to cap to give a bearing for the boards. The floor of the room is covered with boards.

After the room is boarded up and the timbers drilled and charged, the drift is breasted up (see plan and elevation Fig. 22). With some men the blasting of a room is a haphazard operation, with others a regular method is followed as shown by the arrows in the plan. At 1, 2, 3, the caps are left across the room, and so act as props and allow the room to fill up without disturbing the timber against the solid ground.

In heavy ground some of the timbers are likely to be missing or the pillar next to the cave may be broken up. Under such conditions it may be better to drop back one set from the caved ground (see Fig. 23 lower sketch) and open up the first tier in the solid, that is remove the entire tier of which *A′* is a unit from floor to back and end-cave to end-cave. This leaves intact the tier which ordinarily is removed first. After boarding up the new tier to be prepared against the caves, the pillar is attacked from the top downward, braced if necessary against the side pressure of the caved ground. As working on this pillar progresses it is frequently found that posts were

Square-set Slice showing set of tools.



Square-set Slice boarded up next to caves.
FIG. 29.

FIG. 30. Square Set Slice boarded up ready to cave.

blasted down with the last cave. When this is the case, these posts are replaced as the ground is timbered.

When the change is made from drift slicing to square-set slicing, a small pillar is always left between the two slices (see Fig. 23 upper sketch). The sets *A* and *A'* are then taken out; the tier *A'* is completed as in the case of heavy ground; next the

FIG. 31. Sill Floor of a Square-Set Slice. The ore from upper set is mucked directly into cars. The timber from the adjoining cave is breaking through.

tier at *A*, the pillar at *X* still remaining; the room is then boarded up; the pillar *X* is taken out and the room finally caved. Succeeding rooms are mined adjacent to the cave in good ground, or in the solid in the case of heavy ground.

Different mining captains have varying methods of caving a worked-out slice. At the Chisholm Mine the center posts of some of the caps between the center posts and the cave are shot down. Small holes are drilled into the above mentioned timbers

and a stick of powder is charged into each hole. The holes are all shot at once and the ground rapidly settles. Others prefer to shoot down the posts next the cave, one or two of the center posts and a few caps. They claim that the cave settles better and that the timber is in better shape to work out the next slice. Any rock encountered in the slice is left there if it is too big to be easily removed. If the

FIG. 32. Square-Set Slice in heavy ground.

rock mass is large, the slice is run up over it, or run under it, the ore surrounding the rock being extracted by a subsequent slice.

The percentage of extraction is high if the work is properly and carefully done. The shift bosses and captains watch this very carefully and make the contractors keep the floor clean, trim the back, and catch all ore on the sides. The extraction varies from 95 to 98 per cent.

OPERATING DETAILS

Contract labor.—All regular breaking (drifting and stoping) is run on contract. A contract gang consists of four men, two on day and two on night shift. The contractors do their own breaking, timbering, mucking, and local tramming. They deliver the ore into chutes from which it is later drawn by company trammers operating either mule or electric haulage, sometimes both. The tramming distance for contractors varies; it ranges from 75 to 300 feet, and sometimes in this distance one or more transfers are made (see Fig. 33) on account of the drift having jumped up a few feet following a sudden rise in the bed-rock. The contract price is adjusted to meet these varying lengths of tram and number of transfers.

The contractor furnishes his own supplies, sets his own timber, and lays his track. The timber is supplied to him underground at a number of convenient points, whence he runs it to his contract on his own time.

The usual list of tools for a contract includes: 4 picks. 4 augers (3½, 4, 6, and 8 feet), 2 pyramid pointed moils of 1¼-inch octagon steel, 3 feet and 6 feet long; 2 scrapers; 2 tamping sticks; 2 Ajax shovels (at $1 per pair); 1 7-pound hammer; 1 saw and 1 axe (at $1 each); 1 powder heater, rented at 50 cents a month; 1 box in which to store powder, fuse, caps, etc.

The contractor is charged with the tools at the prices given above. Picks and augers are supplied, kept in repair, and sharpened by the company. The contractor is charged $6 for a box of 240 Granite candles and $8 per 50-pound box of Red Cross dynamite.

Contract prices are set by the mine captain who must give careful consideration to such questions as thickness of ore, hardness, intrusions of rock, length of tram and number of transfers, etc. The high prices charged for powder is taken into consideration in setting the contract price. If the miner uses excessive powder charges he is the sufferer. Contractors are usually allowed to pick their own partners. The time-keepers watch contracts carefully and absences on either shift are recorded. The absentees are docked ¼ to ½ shift for tardiness or early quitting and this is taken into account when distributing the contract earnings for the month. The distribution is made by the company and not by the individual. Under ordinary operating conditions a double slice 16 feet wide, 15 feet high, 50 feet long is taken out in 2 weeks by a 4-man gang. The usual rate of advance for a single slice, one set wide, is from 150 to 175 feet per month.

Drift contracts are let by the foot. "Slicing" contracts are usually let by the car of a given capacity gauged, not by the local trammer's cars, but by the number of cars drawn from the chutes on the tramming levels by the company trammers. Sometimes, when several contracts dump into the same chute, the contract price is set per running foot of drift-slice for a given height of slice. In square-set slicing the price would be gauged per set or per car.

Drifts.—Horizontal openings, whenever possible, are run in the ore on account of the cheapness and speed with which they may be driven. The Missabe mines are rarely equipped with machine drills, hence cross-cuts in rock are avoided. The main

ulage levels or drifts are usually timbered with a three-piece set of unframed tim-r; 9-foot posts and 10-foot cap; inside timber dimensions 8-9 feet. The post is in a hitch in the floor without sills, floor-plates, or mud-braces. The sets are held in ace by four 3-inch sprags, two between the caps and two between the posts. The gging is either 3 to 4 inches round or 6 inches sawed. It takes 2 men about 1½ urs to put in a set. In smaller drifts the posts are completely set in hitches cut into e walls. The ore is soft and the holes are put in with an augur or gad. A round nsists of from 5 to 7 holes, 6 feet deep. A 6-foot hole is drilled in from 10 to 30 nutes. The back holes are loaded with from 7 to 10 sticks of 40 per cent Red oss and the lifters with 6 sticks. The upper holes are fired and mucked first and e lifters afterwards. This reduces the powder cost, but entails a wait twice in a

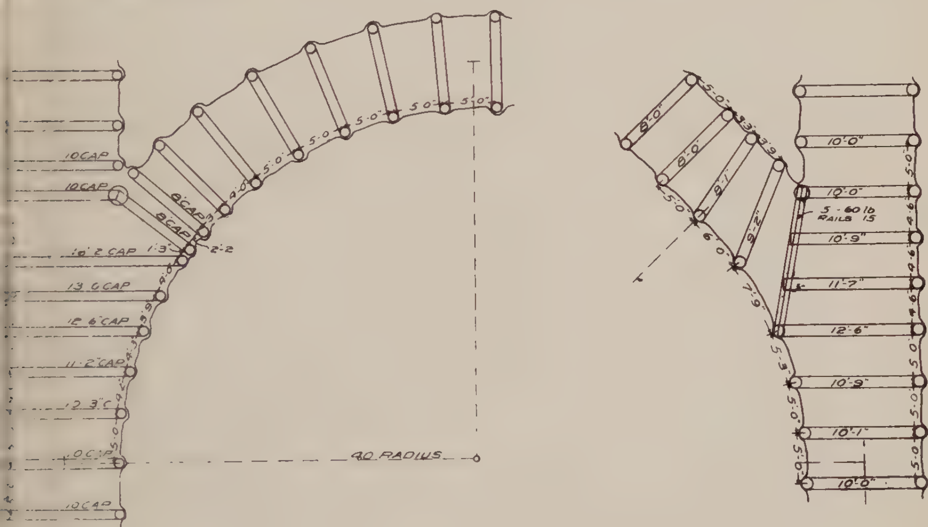


Fig. 34. Standard Curves.

ound for the smoke to clear. So in a drift where the ventilation is poor the entire ound is fired at once, more powder being used.

The average monthly progress on main haulage drifts of this size ranges from 0 to 125 feet a month and the contract price ranges from $3 to $4 per foot, includ-g cost of timbering. The rate on very wet ground or ground containing many layers f taconite sometimes runs as high as $5 to $6.

Rock drifts are usually run by hand, the rate of advance being 25 to 40 feet per onth. Some mines in which there is much rock work to be done are equipped with ompressor plants. The speed of driving will then range from 50 to 75 and some-imes 100 feet a month. The Missabe miner is essentially a soft-ground miner and oes not know how to handle hard ground to advantage. Contract prices on rock rifts range from $9 to $12 per foot and sometimes up to $20. Fig. 34 shows the mbering of a 90° turn with a 40-foot radius; all timbers are cut to exact size and

carefully lined in from transit points. The right-hand sketch shows a sharper turn and a variation in the method of support; a special "taking-up" cap, consisting of several lengths of 60-pound rail. The rate of advance in the smaller drifts ranges from 200 to 225 feet per month on a four-man contract.

At the Corsica Mine a piece of heavy ground was encountered that necessitated spiling. Six-inch round poles, 8 feet long, were sharpened and driven over a false set, about 12 poles to a set. The contract labor price was about $2 and the cost of timber (sets at 4-foot centers) and lagging about $1.50 per foot, making a total price of $3.50 per foot, the complete cost of a spiled drift.

FIG. 35. **Drift-Set, Chute and Mine Car, Fayal Mine.**

Generally speaking the ore is sufficiently firm so that small and even fair-sized drifts will hold up several years without timber. The ore, however, is the reverse of compact and slowly consolidates, creeping towards the openings and in some cases throwing much weight on the timbers. For this reason main level timbers are very heavy. There is frequent displacement and crushing of timbers in many of the mines, and drift repair work is constantly going on. The older the mine the heavier the repairs. Often shipments are curtailed from a mine designed for a certain tonnage; the timbers then take weight and much repair work necessarily follows. Two men can take out and replace from 1½ to 3 sets of timber in a 10-hour shift for large and small drifts respectively. Paint-rock drifts when wet must be retimbered

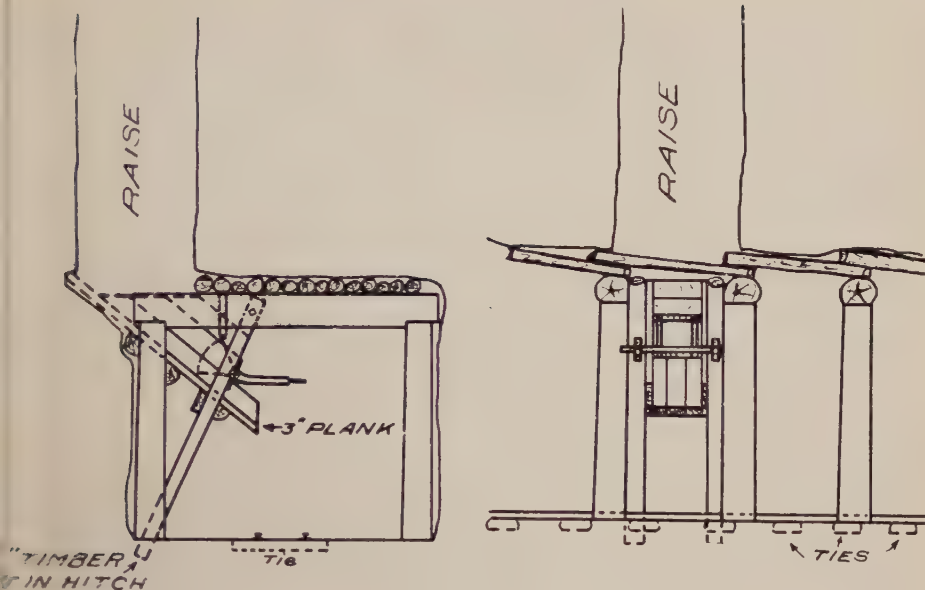


FIG. 35a. Standard Chute.

o or three times a year. One mine now worked on a large scale was formerly orked in a haphazard way. There are many drifts to maintain. Often retimbered round will not hold up over one week. A repair crew of from 10 to 15 men is quired in addition to the frequent repair work done by contractors who receive regu-r wages when on this work. Repair work is sometimes contracted for at $1 to $1.25 er foot for 8 by 7-foot drifts. This arrangement proves to be cheaper for the com-any and at the same time the timber man draws higher wages.

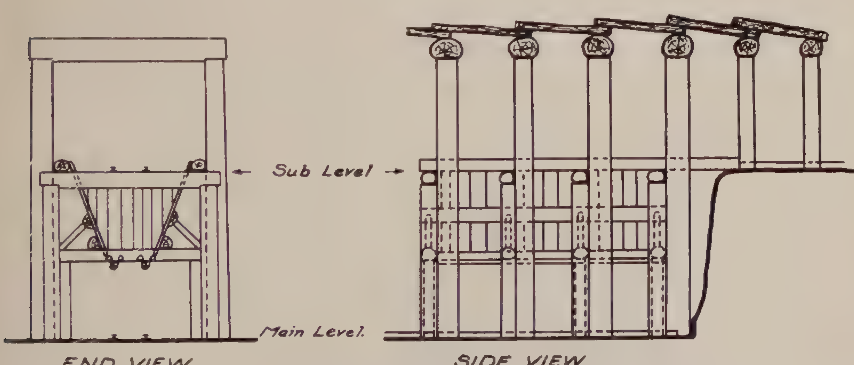


FIG. 36. Hanging Chutes.

Raises and chutes.—Raises vary from 4 by 5 feet to 5 by 8 feet, the variation d pending upon the purpose they are to serve. They are contracted for at $1.50 to $3.5 the rate depending upon the ground and the method of timbering. Raises for the tran fer of ore from the sub to the main tramming levels are usually 4 by 6 feet. They te minate in ore chutes fitted with quarter pan gates as shown in Figs. 35 and 35a. On t sub next to the bottom there is no room to put in the ordinary chute and in the Chishol district hanging chutes are used (see Fig. 36). The bottom of the chute is made loose boards resting on rails or I-beams that run along the length of the chute ar carry the major part of the weight. They are often 15 to 20 feet in length and ho a number of cars of ore. One or more cars may be filled at a time by simply pryin open and removing a few of the bottom boards. Fig. 37 illustrates one form chute used in square-set slicing while the photograph, Fig. 31 on page 56, shov a simpler form.

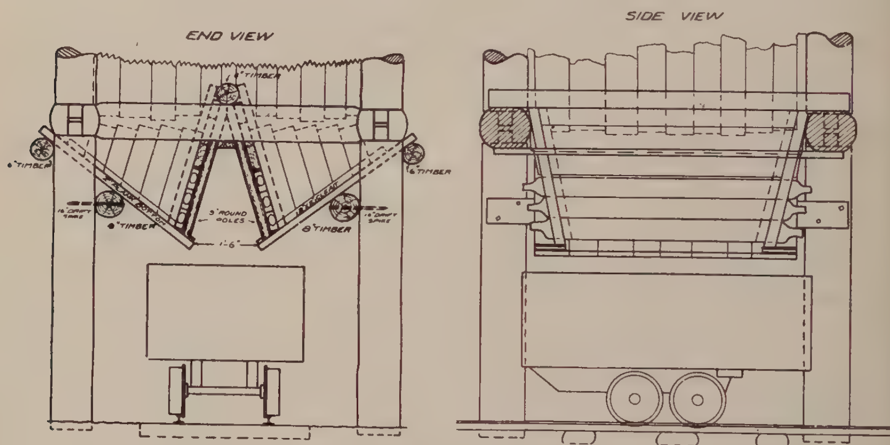


Fig. 37. Square-Set Chutes.

Drilling.—Holes are drilled with augers or gads. An auger hole averages 1 minutes in good ground. When a hard bunch of rock is encountered, the auger removed and the moil driven in 5 or 6 inches. After the hard strata is cracked up ar passed through, the auger may again be used.

In a 7 by 12-foot breast five 6-foot holes are put in, one in the back, two brea holes and two lifters. Four sticks of 1⅛-inch dynamite are charged to the lifters, 5 the back holes and 7 to 9 sticks to the breast hole; 3½- to 4-foot fuses are use The three upper holes are shot first and after mucking out, the lifters are fired.

Mucking and tramming.—The miners use round-pointed, short-handled shove and muck from a 1-inch plank floor into two- or three-ton cars. They tram the o to a chute and dump. Length of tram in slices is on the average up to 300 feet. Th main tramming levels are usually equipped with electric haulage systems.

Mine timber and timbering.—Most of the mine timber used on the Missabe Range is cut within such a radius that it is hauled in during the winter time. It is principally pine and tamarack. The cost of mine timber, delivered in the timber yard, ranges from 4½ to 5½ cents per running foot for pine and tamarack up to 10 inches diameter at the small end, cut in 16-foot lengths. The timber used for heavy drift sets and for square-set work, when bought separately, costs from 8 cents up to 15, and sometimes 20 cents per foot, the cost depending on the diameter and market conditions. Large purchases of ordinary timber usually contain quite a proportion of the heavier timbers (12- to 15-inch), so these need seldom be purchased separately at the higher price. Lagging costs from 1 to 2 cents per piece or $5 per cord piled in the yard. A cord will serve for 200 to 250 tons of ore. No. 5 plank, used for boarding up the rooms, costing $7.50 per thousand, is charged up at $10 per thousand to allow for waste. A drift slice 50 feet long, 2 sets wide, with 10-foot posts, will yield on an average 600 tons and require roughly about 2,000 board feet for flooring and boarding, between 3 and 4 board feet per ton of ore.

Some mines frame their timbers by day's pay and others by the piece, the contractor making from $2.50 to $3.00 a day. Current contract rates are as follows:

Three-piece drift set, 7 to 10 cents each.

Square-set posts, top and bottom respectively, 12 and 9 cents each.

Square-set caps, 7 cents each.

Ties are made of 6-inch lagging, being squared top and bottom and cut into 6-foot lengths, at a cost of 2½ cents each. They are cut in two underground.

On all contract work, whether drifting or slicing, the miners set up their own timbers, which are furnished, already framed, by the company and stored at some convenient place in the mine. The framing is comparatively simple and consists of cutting a flat-bearing surface at the ends of the caps where they rest on the posts. Caps are usually 7 feet long, of 6- to 9-inch round timber. Posts range from 7 to 14 feet long and of same diameter as the caps. Their length depends on the height of the slice and this is, of course, determined by the height of ore above the level of the haulage drift floor. Sprags are made of 3- to 4-inch timber. For lagging, 3-inch poles or 6-inch split poles are used.

Sprags are put in as shown in Fig. 19, Plate IV, between the tops of the posts and the end of the caps. They are cut to exact size and pounded into place. Occasionally when one of the posts next the cave cracks, it is necessary to put a sprag to the cracked post from one of the center posts.

Caps are uniformly 7 feet long, so it requires two caps, each supported by two posts, to reach across a slice. There are, then, four posts, one next the cave, one next the ore pillar, and two center posts touching each other. The caps have blocks between their ends so as to have the effect of a solid cap taking the pressure from the cave and transmitting it against the solid ore pillar. Rough blocks are used to block the caps against the rock. Wedges are cut by hand from old blocks and are rather crude.

Pole and split lagging is used overhead and on the sides where necessary. When a timber mat has once been formed, it is not necessary to use much timber overhead.

A few large stringers put across the caps will hold it up. One-inch plank is nailed on the posts next to the rock after a slice has been worked out so as to hold the caved material in place after caving.

Variations.—At the old Leonard Mine, Chisholm district, the ore runs from 18 to 21 feet high, almost too low for two drift slices, so a combination drift and square-set slice is used. A single drift slice is taken out next to the cave, using a 12-foot post.

FIG. 39. A Prop-Slice.

At the last drift set an upraise is made through the ore to the capping above and a square-set put in ranging from 4 to 7 feet high. This square-set slice is continued over the drift set, the length of post being varied to suit the capping. A second slice is then taken out beside the first, starting at the end, carrying combination drift and square-set slice full height. The four sets on the other side of the en[illegible] are then taken out full height in one operation.

At times the ore-bodies dwindle down to a thickness of 5 to 18 feet contained between two layers of taconite, the roof layer of taconite being 40 to 50 feet thick, very hard and tough and breaking or caving only when a large area is mined out. Such ground is developed by a main haulage road from the shaft, from which drifts are run at 50-foot centers to the shore line where a turn is made and the pillar sliced out in slices 10 feet wide and 50 feet long. The drifts and cross-cuts are timbered with regulation 3-piece sets. The slice, however, is timbered with a row of props (see Figs. 38 and 39), placed on the rock or cave side. These props range from 10 to 18 inches in diameter; they are placed from 3 to 6 feet apart according to the ground.

Where the ore backs exceed 12 feet, square-set timbering is used one set high with two props of the necessary height superimposed upon each cap. Prop-slicing leaves large open rooms which are not caved unless the roof requires it. The roof is hard and stands for a long time, finally it begins to crack, and when it caves comes in large blocks. Prop-slice ore is usually considered quite cheap and the timber cost is low. On a 12- to 16-foot slice two men will get out from 8 to 14 cars a day and the contract price runs about 70 cents per car (supposedly 3 tons). For square-set work with prop tops the price runs about 5 cents less per car. When the ore has dwindled down to 5 feet, it is usually very hard and the output for two men runs as low as 4 to 5 cars, in which case the contract price may run to $1.40 or $1.50 per car.

SURFACE PLANT

The surface plant for the better class of Missabe underground mines, designed for a life of 10 to 20 years and a monthly output of 25,000 tons or more, would comprise the following buildings and equipment:

Steel headframe, stock-pile trestle and electric haulage for same Cost from	$10,000	to	$12,500
Engine and boiler house equipment as follows— 3 150 h. p. boilers; 1 duplex 16x20 double drum reversible hoist; 1 14x24 Corliss engine; 1 100 k. w. generator; 1 small compressor; 1 concrete smoke-stack. Cost from	20,000	to	30,000
Coal dock. Cost from	2,000	to	3,500
Water tower. Cost from	1,000	to	1,500
Miners' dry for 250 men. Cost from	7,500	to	8,500
Shops and equipment (machine, blacksmith, and carpenter). Cost from	10,000	to	20,000
Warehouse and sheds (carrying a $10,000 stock of supplies). Cost from	2,500	to	4,000
Office and equipment. Cost from	5,000	to	10,000
	$58,000	to	$90,000

The total cost of buildings and plant will run from $60,000 to $90,000 according to the construction. This is exclusive of the "location," which in the case of a property remote from a town will cost from $20,000 to $75,000, according to the requirements. A miners' boarding house may cost from $5,000 upward. Cottages, 4-room, 6-room, and 8-room, respectively $750, $1,000, and $1,500. Superintendent's house, $2,500 to $4,000.

The cost of a 250-foot shaft with station and pocket complete under ordinary conditions will approximate $25,000. Cost of pump room and 1,500-gallon pump, $15,000. Cost of underground electric haulage, including track and cars, $15,000. Total cost of shaft and underground equipment, $55,000.

The total cost of surface plant and equipment, together with shaft and underground equipment, will run from $125,000 to $150,000, when close enough to town to eliminate quarters for the men; while a property remote from town will require an additional $30,000 to $50,000 and possibly $75,000 for employees' living quarters.

DEVELOPMENT CREW

The crew will vary from 75 to 100 men during shaft-sinking and surface construction work, which will consume from 8 to 12 months according to conditions.

Thereafter, during development work, the crew will run from 100 to 200 men according to the desired speed of development and the number of working faces available.

The amount of development necessary before a mine is ready to commence stoping or slicing on any scale varies with circumstances from 10,000 to 30,000 feet and possibly more. Development work is practically continuous until the decline of the mine.

A typical Missabe mine, during the height of its production, after slicing is in full swing, will employ from 175 to 250 men according to the tonnage mined, divided as follows for a monthly output of 20,000 and 30,000 tons respectively.

Contract miners—slicing and development	100 to 150
Company account men—underground	30 to 50
Surface crew	40 to 50

The time required for development will vary with circumstances from 15 to 18 months unless hampered by sinking difficulties, construction delays, and other untoward conditions.

TYPICAL UNDERGROUND MINES

The Harold Mine.—This mine, situated 3 miles from Hibbing, may be cited as a typical Missabe underground mine offering no operating difficulties. The entire tract (40 acres) is underlaid by a fairly good grade of non-Bessemer ore with some rock admixture. The thickness of the ore varies from 16 feet to 60 feet, average thickness practically 40 feet. The tonnage estimated from the drill records is between 4 and 5 million. About two-thirds of the area is at present developed through the Harold shaft by one main level 3,490 feet long and 5 sub-levels with an aggregate

SUMMARY OF HAROLD MINE DEVELOPMENT

Dec. 1, 1909, to June 1, 1910	Shaft sinking and surface construction. Average number of men employed 76, minimum 24, maximum 104.
June 1, 1910, to Oct. 1, 1910 [4 months]	Construction work. Station and pocket. Development work, 543 ft. progress on Main Level and 1,368 ft. on Sub-level. Average number of men employed 92, minimum 83, maximum 105.

	Development Footage	Development Tonnage	Slicing Tonnage	Average Number of Men Employed: Underground (U. G.): Development	Underground (U. G.): Slicing	Underground (U. G.): Company Account	Underground (U. G.): Total (U. G.)	Surface	Total U. G. and Surface
Oct. 1, 1910, to March 1, 1911 [5 months]	17,000	70,000	....	Av. 92	....	Av. 32	Av. 124	Av. 55	Av. 179
				Min. 72	....	Min. 11	Min. 83	Min. 44	Min. 127
	....	....	....	Max. 105	....	Max. 47	Max. 152	Max. 65	Max. 198
March 1, 1911, to Jan. 1, 1912	16,123	53,434	214,356	Av. 30	Av. 70	Av. 56.5	Av. 156.5	Av. 50	Av. 206.5
				Min. 4.2	Min. 35	Min. 48	Min. 87	Min. 41	Min. 128
				Max. 71	Max. 87	Max. 65	Max. 223	Max. 68	Max. 273

During 5 months development period ending March 1st, 1911, the following averages obtained:
1.42 ft. advance and 5.83 tons per miner per shift.
1.05 ft. advance and 4.35 tons per man underground per shift.
0.72 ft. advance and 3.00 tons per man employed (underground and surface) per shift.

During 10 months continued development and slicing period ending Jan. 1, 1912, there was mined a total of 267,790 tons, 80% of which was slicing ore; the following averages obtained:
2.07 ft. advance per miner on development work per shift.
10.03 tons of ore per miner slicing per shift.
6.58 tons of ore per man underground.
4.99 tons of ore per man employed underground and surface.

length of 31,797 feet (January 1, 1912). The tonnage developed by this work is approximately 2 million tons.

Development.—In fourteen months the mine was developed to the slicing stage. The average crew during this period was 105 men, on two shifts. The total number of man-shifts worked was 38,220. This includes all surface construction labor, excepting steel head-frame which was built on contract. This property is operated by the Oliver Iron Mining Co. On account of its proximity to the Hibbing district headquarters the Harold plant is quite simple, and the labor chargeable to construction is much below the average. Through the courtesy of the Oliver Iron Mining Co. I am enabled to publish the table of operations on the accompanying insert, from which the brief summary on page 67 is drawn.

The Whiteside Mine.—This mine is one of a group of properties operated by the Shenango Furnace Co.; it is situated at Buhl, some 6 miles by wagon road from the company's headquarters in the Chisholm district. From an operating standpoint it may be considered as a typical wet underground mine.

Over 35 acres of the 40-acre tract comprising the Whiteside is underlaid by ore. The surface contour varies from 915 to 935 feet; the top contour from 810 to 875 feet and the bottom ore-contour from 659 to 785 feet. The depth of overburden therefore varies from 50 to 125 feet. The thickness of ore varies from 200 feet in the southeast corner to 90 feet in the center and perhaps 45 feet in the northwest corner. The main level is opened at the 730-foot contour and is underlaid by from 10 to 80 feet of ore in the central and southeast portions of the tract respectively.

The ore-body consists of a number of irregular layers of Bessemer, non-Bessemer, and lean ore, interbedded with layers of taconite. The ore runs high in moisture, averages around 55 per cent iron and is uniformly low grade so that there is little high grade ore to help bring the poorest ore to grade. The tonnage developed is variously estimated at between 5 and 6 million tons. Operations were started in the woods, with deep snow on the ground, late in the winter of 1909. The location at the time was a mile or so from the railroad. All materials, supplies, and machinery for the mine, the surface plant, and quite a large location were hauled in over abandoned corduroy roads. When the mine was ready to ship, the railroad built a spur to the shaft. Fig. 40 shows the completed surface plant.

During the sinking of the shaft the water was handled by three to four Prescott 14x8x24 sinking pumps, re-enforced by Camerons when the maximum of 1,800 gallons was reached. Upon completion of the shaft these Prescott pumps were temporarily used in a horizontal position during the cutting of the pump room and the installation of a triple expansion pump of 3,000-gallon capacity.

During the shaft-sinking period the daily average number of men employed varied from 26 to 30 men on mining work and a varying number on construction. The shaft was sunk to within a few feet of its ultimate depth by the end of September. Sinking was then stopped pending the construction of a permanent boiler plant. The shaft was completed during February, 1910.

HAROLD MINE

STATEMENT OF OPERATIONS AT HAROLD MINE SHOWING AVERAGE NUMBER OF MEN EMPLOYED, DEVELOPMENT AND PRODUCTION

FROM DECEMBER 1, 1909, TO JANUARY 1, 1912

Month	Average No. Men Employed up to Commencement of Slicing, February 1, 1911				Average No. Men Employed from Commencement of Slicing on February 1, 1911, to January 1, 1912					Development [Lineal Feet]						Ore-Production			Water
	Surface*	Underground		Total	Surface	Underground			Total	Sinking Shaft	Raising	Cross-Cuts	Drifting			Development	Slicing	Total	Gallons Per Minute
		Miners	Others			Slicing	Development	Company Account					Main-Level	Sub-Levels	Total Development				
1909																			
December	24.3			24.3															
1910																			
January	57.7		5.0	62.7						29.0									100
February	70.4		7.8	78.2						33.0									100
March	78.3		8.1	86.4						45.0									100
April	94.6		9.6	104.2						38.0									400
May	87.6		12.9	100.5						13.0									450
June	77.5	6.7	8.8	93.0						2.5			40		40	735		735	500
July	63.1	14.8	5.1	83.0									65		65	741		741	500
August	58.2	20.6	7.0	85.8									142		142	2,123		2,123	500
September	46.0	50.9	8.3	105.2									296	1,368	1,664	7,375		7,375	500
October	44.1	72.8	10.9	127.8							35	255	395	2,456	2,851	10,639		10,639	500
November	44.7	89.2	11.1	145.0							443	337	479	2,468	2,947	14,520		14,520	500
December	51.3	90.0	24.7	166.0							356	180	488	2,966	3,454	14,491		14,491	500
1911																			
January	56.3	105.0	36.2	197.5							356	451	406	4,097	4,503	16,788		16,788	500
February					63.3	13.7	74.2	47.1	198.3		307	131	478	3,020	3,498	13,349	2,697	16,046	500
March					65.1	34.5	71.1	60.0	230.7		441	250	437	2,746	3,183	10,068	14,342	24,410	500
April					51.2	46.3	63.2	65.5	226.2		468	235	264	2,566	2,830	9,858	12,416	22,274	460
May					36.6	61.7	54.1	60.2	212.6		442	244		2,867	2,867	8,494	21,551	30,045	420
June					46.3	70.1	38.0	57.5	211.9		203	363		2,266	2,266	6,978	21,867	28,845	380
July					44.8	79.0	24.1	55.9	203.8		279	338		966	966	3,436	25,940	29,376	350
August					40.7	87.3	10.7	56.3	195.0		129			530	530	1,913	29,058	30,971	350
September					49.8	77.8	8.6	52.3	188.5		60			536	536	1,793	23,488	25,281	350
October					43.7	79.3	4.2	49.8	177.0			15		264	264	809	24,096	24,905	350
November					47.5	78.7	18.1	59.5	203.8		68	137		875	875	2,898	21,084	23,982	350
December					68.7	76.0	44.0	48.0	236.7		206	925		1,806	1,806	7,087	17,817	24,904	350
Total										160.5	3,793	3,861	3,490	31,797	35,287	134,095	214,356	348,451	

NOTE—*Surface labor in first column includes clearing site, building construction, machinery installation, except head-frame.

The normal shaft crew consisted on an average of:

	Day shift	Night shift
Foreman	1	1
Shaftmen	5–6	5–6
Surfacemen	1	1
Fireman	1	1
Engineer	1	1
Pumpmen	2	1
Blacksmith	2	0
Carpenter	1	0
Landers	1	1
Teamsters	1	0

A total of 16 to 17 men on day shift, and 11 to 12 men on night shift.

This accounts for 754 man-shifts and a payroll of approximately $1,800. Most of the construction carried on during this time was contract work and the total construction account is itemized on page 72.

Underground work proper began in March with about 50 men underground and 75 men all told. The progress of development work, together with the men employed, payroll, supply account, and ore shipments, may be followed by studying the accompanying tables. The development work to date, January 1, 1912, comprises:

Main Level—

5,300 feet with 8 headings to be extended an aggregrate distance of 2,400 feet.

Sub-Level—

186-foot sub; opened August, 1910; ore area roughly 30 acres; 10,600 feet of drifts with 1,500 feet to complete.

166-foot sub; opened August, 1911; ore area 420,000 square feet; 5,100 feet of drifts with 100 feet to go.

142-foot sub; opened November, 1911; 700 feet of drifts with 1,000 feet to go.

127-foot sub; opened October, 1911; ore area 120,000 square feet; 1,200 feet of drifts with 500 feet to go.

Including the three timber shafts there is an aggregate of 2,000 feet of raises. The total development footage exclusive of main shaft and stations is roughly 28,000 feet, with 6,500 feet estimated as necessary to complete.

Ore production from the development work began in March, 1910, with 1,791 tons. During March, April, and May, 1911, several thousand tons were mined from the centers of 8 by 10-foot pillars on the main level by opening out a sill floor, 3 sets wide and 11 sets long. Rooms were then run up 4 to 6 sets high past the 186-foot sub to the solid rock capping above. The object was twofold: To get a production of high-grade ore and to drain quickly the wet ore-body. The results were very satisfactory. This high-grade ore, containing 60 per cent iron or better, was mixed on the stock pile to bring up to grade the development ore which at that time contained from 45 to 49 per cent iron. The total development tonnage is approximately 160,000 tons.

Slicing was begun in the center of the 186-foot sub in April, 1911.

From 18 to 20 points of attack were available, but only 14 slices were started because the iron ore market would not absorb the low-grade ore. As the work of slicing is extended to the various subs it will be possible to maintain about 30 gangs with a production of 30,000 tons per month of 26 days.

An electric haulage system was installed on the main level and a secondary system on the 186-foot sub-level to handle the ore from the northwest corner. The aggregate length of haulages amounts to 7,200 feet. The train on the sub dumps direct into an inclined chute that discharges into the main-level shaft pocket.

All machine, blacksmith, and carpenter work for the various mines operated by the Shenango Furnace Co. is done at the central shops at the Shenango Mine. The shops represent an investment of $38,000, about $20,000 for the building and $18,000 for the equipment. The monthly operating cost of these shops is about $2,500. This includes shop labor and supplies such as coal, waste, oil, etc., but of course does not include shop material.

There is also a large fire-proof central warehouse with a $30,000 to $40,000 stock. A perpetual inventory system simplifies the warehouse bookkeeping. The local warehouse at the Whiteside carries about $2.500 in stock. The company charged all operations to a "development and construction" account until March, 1910, when ore production began and a "cost sheet" was started. All subsequent expenses have been charged to ore production, except such labor and supply items as were properly chargeable to construction unfinished. The latter were charged to the construction and development account which, therefore, includes all preliminary expenses incident to opening the mine and all construction and equipment. Including the contract price of some $12,000 for houses now building, there has been charged to this account a total in round figures of $210,000.

Allowing the mine 12 years' life on an annual production of 400,000 tons and charging the construction account with 6 per cent interest, a charge of 6 cents per ton of ore will wipe out the construction account.

Operating expenses to January 1, 1912, added to the above-mentioned construction charge bring the total gross expenditures to date in round numbers to $500,000. The mine should be credited with $195,000, the value of 129,605 tons shipped during the 1911 shipping season with a value on cars at the mine of approximately $1.50 per ton (based on 1911 ore prices, rail and lake charges).

FIG. 40. Surface Plant, Whiteside Mine.

In addition to the ore shipped there is almost an equal tonnage on the stock-pile. Following is an itemized statement of the plant and equipment, both surface and underground:

Cost of shaft complete (steel lined), depth 249 feet		$37,749.23	
Station and pocket, and rock drifting		11,719.73	
Concrete pump room and pumps, condensers, steam and water lines		23,333.67	
Electric haulage—			
3 locomotives	$6,059.08		
18 train cars (3¼-ton)	2,960.00		
20 sub-cars (1-ton)	670.00		
Rolling stock	$9,689.08		
Ties, rails, and angle plates, for 7,200 feet of track	4,396.07		
Trolley-wire and hangers	1,075.04		
Armature	271.38		
Labor and supplies, etc.	1,958.60		
Total track, wire, and miscellaneous supplies and construction	$7,701.09		
Total electric haulage		17,400.17	
TOTAL SHAFT AND UNDERGROUND EQUIPMENT (exclusive of development)			$90,202.80
Head-frame and trestle	$12,613.76		
Stock-pile electric haulage equipment	4,391.06		
Total head-frame, trestle, and stock-pile motor equipment		$17,004.82	
Total engine and boiler plant		34,353.41	
Partial cost of equipment—			
Three 72x18 150 h. p. horizontal tubular boilers	$2,261.38		
Duplex 16x20 reversible double drum hoist	6,970.00		
100 k. w. generator and 14x20 Corliss engine	3,900.00		
Concrete smoke stack	2,401.69		
Water tower		1,680.11	
Coal dock		4,513.47	
Office and warehouse		2,806.46	
Office furniture and equipment		746.09	
Miners dry (258 lockers)		8,072.99	
Pipe-house		1,000.00	
Location houses and boarding houses (29)		37,714.66	
TOTAL SURFACE BUILDINGS AND PLANT			107,892.01
TOTAL CONSTRUCTION ACCOUNT, SURFACE AND UNDERGROUND			$198,094.81

CLASSIFIED LABOR STATEMENT

	Daily Av.		Man-shifts per month	
Contract Miners		90.0		2,352.25
Company Account Labor		59.5		1,542.85
Total Labor		149.5		3,895.10
Subdivision of Company Account Labor				
Transportation—				
Pocketmen	2.1		54.00	
Ditchmen	2.4		62.75	
Trackmen	1.4		36.00	
Motormen	4.0		101.00	
Brakemen	4.0		101.00	
Timber Trammers	2.0		52.00	
Timber Landers	2.0		52.00	
Sub-level				
Tram and Tally Boys	1.2		31.50	
Skiptenders	2.0		20.00	
Total Transportation		21.1		542.25
Pumping—				
Pumpmen	2.2		57.00	
Pipemen	3.2		82.00	
Pump and Pipe Boss	1.0		26.00	
Total Pumping		6.4		165.00
Surface Labor—				
Engineers	2.25		58.5	
Firemen	2.40		62.0	
Pocketmen	3.40		88.4	
Electrician	1.50		38.8	
Timber Framers	1.65		43.0	
Carpenters	1.40		36.2	
Blacksmith	1.00		26.0	
General Labor	5.00		130.7	
Mine Patrol	2.00		52.0	
Dry-man	1.00		26.0	
Teaming	1.60		42.5	
Total Surface		23.20		502.1
Secondary Shaft (9 days' operation)				122.0

During the month of December, 1911, thirteen slicing contracts produced 11,310 tons of ore in 1,094.5 man-shifts, giving a tonnage per man slicing of 10.34 tons.

Analyzing the operations at this property the following points of interest may be summarized:

For the Year:	1910	1911
Total number of man-shifts per year	30,015.00	46,072.00
Average number of men employed daily	115.00	150.00
Average monthly pay-roll	$ 6,881.50	$ 10,033.00
Average monthly supply account	3,912.00	5,312.00
Average mine expense for the year (exclusive of general equipment and general expense)	129,526.00	184,141.00

The total man-shifts employed to January 1, 1912	82,213
Total man-shifts to commencement of slicing	42,827
Total development footage to above date	17,617
Total development footage to January 1, 1912	34,678
Total development tonnage	159,369
Total tonnage mined to January 1, 1912	257,566

In 1911 the following supplies were used in mining 207,707 tons of ore, about half of which was development ore:

19,400	lbs. 40% and 95,400 lbs. 27% dynamite, cost	$10,807.17
240,000	ft. of fuse, which, with caps, cost	500.00
201,470	lineal feet of timber, cost	8,497.86
519	cords of lagging, cost	2,620.12
594	tons of coal @ $3.95 per ton, cost	18,146.50

WHITESIDE MINE

CLASSIFIED LABOR SHEET APRIL 1, 1910, TO JANUARY 1, 1912

	1910 April	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	1911 Jan.	Feb.	March	April	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	Average for the Year 1911	
																						Man-shifts Per Month	Av. Men Per Day
UNDERGROUND—																							
Mining—																							
Contract miners	349.00	669.00	875.25	655.75	425.50	....	1,225.00	1,688.75	2,304.50	2,055.50	1,912.25	1,878.00	1,666.25	2,085.25	2,050.50	2,238.00	2,266.50	2,156.50	2,352.25	2,396.50	2,276.00	2,111.13	81.20
Company account miners	47.50	246.90	324.20	1,003.25	275.75	446.25	385.50	418.50	173.70	275.50	137.50	239.75	324.70	251.00	210.00	456.10	221.00	205.00	105.00	73.25	182.25	223.42	8.60
Timbermen	33.50	32.30	49.60	61.75	54.75	55.20	12.00	16.50	36.75	24.00	32.25	79.25	50.00	50.00	50.00	50.50	49.00	53.00	61.75	119.00	83.50	58.52	2.25
Shift bosses	64.50	66.50	64.80	66.50	65.00	58.00	62.25	59.50	58.25	64.00	55.50	61.50	59.00	61.00	54.00	50.00	54.00	52.00	52.00	52.00	50.00	55.42	2.10
Total mining	494.50	1,014.70	1,313.85	1,787.25	821.00	559.45	1,684.75	2,183.25	2,573.20	2,419.00	2,137.50	2,258.50	2,099.95	2,447.25	2,364.50	2,794.60	2,590.50	2,466.50	2,571.00	2,640.70	2,591.75	2,448.48	94.15
Transportation and Hoisting—																							
Loaders and pocketmen	....	....	....	....	....	....	96.25	106.50	142.00	115.50	50.00	54.00	50.00	54.00	52.00	50.00	54.00	52.00	52.00	52.00	50.00	57.13	2.20
Ditchmen	....	....	18.10	12.50	7.50	70.25	6.00	37.00	70.00	56.00	69.25	70.50	58.75	84.00	72.25	61.00	64.00	47.30	62.75	74.00	83.00	66.90	2.60
Trackmen	....	....	....	26.75	88.50	326.75	33.00	44.50	56.25	55.00	60.00	50.00	23.00	47.50	39.50	44.00	45.50	68.50	36.00	44.50	47.00	46.71	1.70
Motormen or trammers	....	....	....	....	....	12.00	54.00	54.00	80.25	97.25	97.50	102.00	101.00	104.50	65.00	50.25	54.00	52.00	101.00	104.00	99.50	85.67	3.30
Brakemen	....	....	....	....	47.00	16.00	54.00	54.50	66.00	89.00	93.00	103.00	99.25	104.50	100.50	92.00	92.00	52.00	101.00	104.00	99.50	94.15	3.50
Timber trammers and landers	....	....	....	....	....	200.60	....	....	37.50	142.25	125.50	105.50	61.50	50.25	52.00	50.00	53.25	52.00	51.50	52.00	50.00	70.48	2.70
Sub-level trammers and tally boys	....	....	....	....	....	....	....	....	....	....	....	....	82.75	160.25	127.50	154.50	231.75	324.30	31.50	30.00	....	142.82	5.50
Skiptenders	51.70	65.90	66.70	68.90	16.60	12.00	52.50	85.00	107.00	104.50	91.75	54.00	50.00	54.25	52.00	49.00	55.00	52.00	52.00	52.00	50.00	59.71	2.30
Total transportation and hoisting	51.70	65.90	84.80	108.15	159.60	637.60	297.75	381.50	559.00	659.50	587.00	539.00	526.25	659.25	560.75	550.75	649.50	700.10	487.75	512.50	529.00	580.11	22.80
Pumping—																							
Pumpmen	99.00	92.80	90.00	82.10	62.00	60.00	62.00	60.00	65.00	62.00	56.00	62.00	60.25	62.00	60.00	62.00	62.00	60.00	57.00	60.00	62.00	60.44	2.30
Pipemen	59.70	77.30	130.20	153.30	134.00	142.90	117.15	98.20	57.20	84.00	55.00	67.70	38.10	62.00	74.50	31.00	53.00	81.50	82.00	37.00	18.00	56.98	2.20
Pump and pipe boss	26.00	33.60	32.50	33.00	31.80	79.90	33.75	32.80	34.00	26.00	38.00	34.00	25.00	27.00	26.00	25.00	27.00	26.00	26.00	26.00	25.00	27.58	1.10
Total pumping	184.70	203.70	252.70	268.40	227.80	282.80	212.90	191.00	156.20	172.00	149.00	163.70	123.35	151.00	160.50	118.00	142.00	167.50	165.00	123.00	105.00	145.00	5.58
TOTAL UNDERGROUND	730.90	1,284.30	1,651.30	2,163.80	1,208.40	1,479.85	2,195.40	2,755.75	3,288.40	3,250.50	2,873.50	2,961.20	2,749.55	3,257.50	3,085.75	3,463.35	3,382.00	3,334.10	3,223.75	3,276.20	3,225.75	3,173.59	122.50
GENERAL SURFACE	1,033.80	1,013.60	867.70	852.80	724.70	508.40	855.50	739.90	783.20	734.30	718.80	769.10	687.30	634.90	578.70	549.50	539.00	513.20	654.35	715.80	706.30	650.10	25.00
SECONDARY SHAFT	470.70	375.20	272.50	131.00	92.00	429.50	124.50	88.00	....	94.25	....	....	....	....	....	....	....	....	46.50	17.00	37.00	48.67	....
TOTAL MAN-SHIFTS	2,235.40	2,683.10	2,773.55	3,147.60	2,361.80	2,417.75	3,203.40	3,583.65	4,088.60	4,010.05	3,677.55	3,740.30	3,436.85	3,892.40	3,664.45	4,012.85	3,921.00	3,893.80	3,895.10	4,017.05	3,911.05	3,839.31	147.60

WHITESIDE MINE

DEVELOPMENT DATA FROM MARCH 1, 1909, TO JANUARY 1, 1912

MONTHLY LABOR STATEMENT IN MAN-SHIFTS PER 24 HOURS

Year	Month	Underground Labor: Mining: Contract Miners	Company Account Miners	Timbermen	Total Miners	Other U. G. Labor: Transportation	Pump and Pipe Men	Total U. G. Men	Surface: General Surface	Secondary Shaft	Total: Total Man Shifts
1909	March	Clearing location									
	April	General surface improvement; work begun on shaft									
	May										
	June										
	July										
	Aug.										
	Sept.	Sinking almost finished; heavy water									
	Oct.										
	Nov.	Construction of new boiler plant; no sinking									
	Dec.										
Total for 1909											5,125.10
1910	Jan.										623.50
	Feb.	Sinking resumed									1,130.20
	Mar.	Development on main level begun									1,767.30
	April	349.00	47.50	33.50	494.50	51.70	184.70	730.90	1,033.80	470.70	2,235.40
	May	669.00	246.90	32.30	1,014.70	65.90	203.70	1,284.30	1,013.60	375.20	2,683.10
	June	875.25	324.20	49.60	1,313.85	84.80	252.70	1,651.30	867.70	272.50	2,773.55
	July	655.75	1,003.25	61.75	1,787.25	108.15	268.40	2,163.80	852.80	131.00	3,147.60
	Aug.	425.50	275.75	54.75	821.00	159.60	227.80	1,208.40	724.70	92.00	2,361.80
	Sept.		446.25	55.20	559.45	637.60	282.80	1,479.85	508.40	429.50	2,417.75
	Oct.	1,225.00	385.50	12.00	1,684.75	297.75	212.90	2,195.40	855.50	124.50	3,203.40
	Nov.	1,688.75	418.50	16.50	2,183.25	381.50	191.00	2,755.75	739.90	88.00	3,583.65
	Dec.	2,304.50	173.70	36.75	2,573.20	559.00	156.20	3,288.40	783.20		4,088.60
Total for 1910		8,192.75	3,321.55	352.35	12,431.95	2,346.00	1,980.20	16,758.10	7,379.60	1,983.40	30,015.85
1911	Jan.	2,055.50	275.50	24.00	2,419.00	659.50	172.00	3,250.50	734.30	94.25	4,010.05
	Feb.	1,912.25	137.50	32.25	2,137.50	587.00	149.00	2,873.50	718.80		3,677.55
	Mar.	1,878.00	239.75	79.25	2,258.50	539.00	163.70	2,961.20	769.10		3,740.30
	April	1,666.25	324.70	50.00	2,099.95	526.25	123.35	2,749.55	687.30		3,436.85
	May	2,085.25	251.00	50.00	2,447.25	659.25	151.00	3,257.50	634.90		3,892.40
	June	2,050.50	210.00	50.00	2,364.50	560.75	160.50	3,085.75	578.70		3,664.45
	July	2,238.00	456.10	50.50	2,794.60	550.75	118.00	3,463.35	549.50		4,012.85
	Aug.	2,266.50	221.00	49.00	2,590.50	649.50	142.00	3,382.00	539.00		3,921.00
	Sept.	2,156.50	205.00	53.00	2,466.50	700.10	167.50	3,334.10	513.20		3,893.80
	Oct.	2,352.25	105.00	61.75	2,571.00	487.75	165.00	3,223.75	654.35	46.50	3,895.10
	Nov.	2,396.50	73.25	119.00	2,640.70	512.50	123.00	3,276.20	715.80	17.00	4,017.05
	Dec.	2,276.00	182.23	83.50	2,590.75	529.00	105.00	3,225.75	706.30	37.00	3,911.05
Total for 1911		25,333.50	2,681.05	702.25	29,380.75	6,961.35	1,740.05	38,083.15	7,801.25	194.75	46,072.45
Total for 3 years											81,213.40

MONTHLY EXPENDITURES, FOOTAGE AND TONNAGE

Year	Month	Expenditures: Payroll	Supplies	Total Expense	Footage: Ft. Advance Per Month	Tonnage: Tons of Ore Mined: Development Ore	Slicing or Stoping Ore	Total Tons	Disposition of Ore: Stock Pile	Shipped
1909	March	$174								
	April	709								
	May	1,285								
	June	1,458								
	July	1,657								
	Aug.	2,317								
	Sept.	2,386								
	Oct.	1,268								
	Nov.	1,087								
	Dec.	1,138								
Total for 1909		$13,479	$7,241	$20,720						
1910	Jan.	$1,639	$1,294	$2,933						
	Feb.	2,972	1,755	4,727						
	Mar.	4,647	3,315	7,962	220	1,791		1,791	1,791	
	April	5,876	4,395	10,271	557	2,964		2,964	2,964	
	May	7,781	5,354	13,135	613	4,809		4,809	4,809	
	June	7,753	5,213	12,966	739	5,077		5,077	5,077	
	July	8,846	5,647	14,493	695	5,569		5,569	5,569	
	Aug.	6,355	2,139	8,494	304	1,831		1,831	1,831	
	Sept.	6,570	1,852	8,422	1,010	990		990	990	
	Oct.	9,211	4,773	13,984	1,716	8,031		8,031	8,031	
	Nov.	10,684	5,243	15,927	2,537	9,115		9,115	9,115	
	Dec.	10,245	5,967	16,212	2,841	9,682		9,682	9,682	
Total for 1910		$82,579	$46,947	$129,526	11,142	49,859		49,859	49,859	
1911	Jan.	$9,851	$5,696	$15,547	2.429	11,632		11,632	11,632	
	Feb.	10,433	5,580	16,013	2,685	12,196		12,196	12,196	
	Mar.	10,402	5,463	15,865	1,361	4,825	8,512	13,337	13,337	
	April	9,412	4,828	14,240	1,667	4,258	8,483	12,741	10,875	1.866
	May	9,553	5,317	14,870	964	7,960	13,461	21,421		19,985
	June	9,792	5,415	15,207	1,175	5,572	13,019	18,591		19,490
	July	10,259	4,050	14,309	1,860	10,887	9,362	20,249		20,249
	Aug.	10,588	5,112	15,700	2,044	11,636	9,323	20.959		20,959
	Sept.	10,040	4,578	14,618	2,242	11,464	6,867	18,331		18,331
	Oct.	9,978	5,798	15,776	2,559	12,067	6,531	18,598		18,598
	Nov.	10,418	5,840	16,258	2,284	8,650	10,964	19,614	9,745	10,139
	Dec.	9,670	6,068	15,738	2,266	8,363	11,675	20,038	20,038	
Total for 1911		$120,396	$63,745	$184,141	23,536	109,510	98,197	207,707	77,823	129,617
Total for 3 years		$216,454	$117,933	$334,387	34,678	159.369	98,197	257,566	127,682	129,617

SPECIAL PROBLEMS

Under this head may be considered the subject of shaft sinking through quicksand and the difficulties incident to mining very wet ground.

SHAFT SINKING

Several of the recognized methods especially applicable to Missabe conditions have been tried. Some were discarded, others modified. The following discussion covers chiefly the principles involved in the various methods in use and gives a detailed description of typical cases.

Missabe shafts are quite shallow. Comparatively little very hard rock is encountered and no deep sinking problems are involved. The main troubles occur in sinking through the surface to the solid rock.

In many localities the surface, while consisting of sand, boulders, clay, etc., is firm and offers no special difficulties. In shaft work the miner usually considers ground "firm" when it can be excavated ahead of the timber or other support. Conversely, when ground must be supported ahead of excavation, it is classed as "soft" or "running" ground. Loose sand, loose gravel, silt, and quicksand belong to this class. So-called firm ground, when it becomes very wet, may lose its firmness. Soft ground, wet ground, or quicksand needs only to be drained to become firm. This is accomplished in various ways according to conditions. It may be a comparatively easy matter or it may be practically impossible within economic limits.

SINKING IN FIRM GROUND

Missabe practice in firm ground does not differ materially from standard methods elsewhere. The excavation is made large enough to allow for timber, blocking, and lagging. The sides are carried down vertically as far as feasible without support. Two trenches are dug (see Fig. 41) paralleling the long dimension of the shaft for the reception of two stringers of 24- to 36-inch timber, 40 to 50 feet long. These stringers are laid at 12- to 15-foot centers and carefully leveled. They serve to support 12 by 12-inch end bearers, 20 feet long, on which a collar-set, having proper inside dimensions of 6 by 18 feet, is placed. The first shaft set is hung by 1¼-inch round hanging bolts from the end-plates of the collar-set. The wall-plate is framed with an upper tenon to rest on a corresponding lower tenon on the end-plate. The dividers that mark the shaft compartments have short tenons resting in corresponding mortices cut in the wall-plates. Practice in this regard varies. Sometimes the end-plates are suspended by hanging bolts from the wall-plates above and the tenons on the wall- and end-plates are reversed.

The sets are held in position by vertical posts or studdles at each corner an at each divider and wall-plate joint. These studdles are not framed. The corne studdles are merely set in half an inch. Divider studdles are cut 3 feet 11 inches they are not gained into the wall-plates, but are held in place by small cleats. Th hanging bolts—2 to an end-plate, 4 to a set—are screwed up tight, putting the studd under compression. The studdles of the first shaft set are shortened and their to fit into ½-inch joggles cut in the under side of the end bearers.

Subsequent sets are hung from end-plates immediately above them. Studdles ar cut 4 feet long, making standard sets 3 feet and 11 inches apart. With 12 by 12-inc timbers the sets are 4 feet and 11 inches center to center. The distance between se may be lessened and the number of hanging bolts increased if the ground is heav

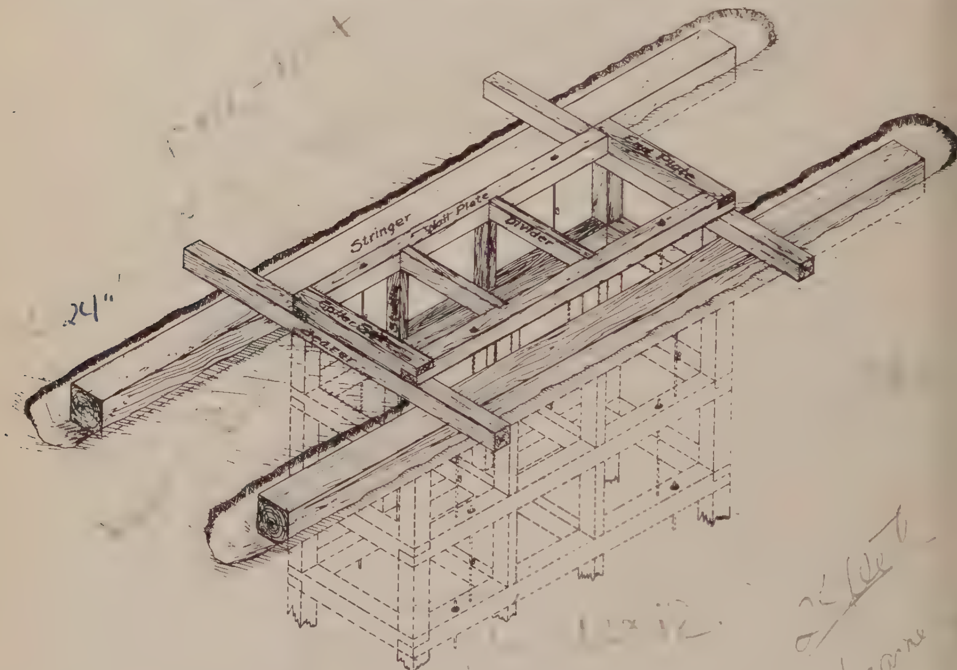


Fig. 41. Collar Set.

The collar-set is usually completed by building up one set high, carefully laggin the outside, and bracing the set to the bearers. The first dirt excavated is banke around the set and carefully tamped to keep out the surface drainage.

Excavation is very easy; there is much pick and bar work and very light blasting The central compartment is used for hoisting and is lined as soon as the dividers are i to prevent the bucket from catching on the timbers.

The shaft is lagged outside the timbers with 2-inch plank in 4 feet 9-inch lengths A 2 by 2-inch cleat is nailed to the wall-plates and end-plates for the lagging to res upon. The lagging is slipped in first at the corner studdles, thence toward the middl

the end- and wall-plates until there is room for one piece only. The key-piece ut to fit this space. It is slipped up past the strips on the upper wall-plate, blocked m behind, and pulled down to its seating on the lower lagging cleat.

As excavation proceeds it becomes necessary to take the weight of the sus-nded sets of timber off from the collar-set and transfer it to bearing timbers in shaft. The interval between bearers depends on the condition of the wall. It ies from 25 to 50 feet and sometimes more. As shown in the sketch (Fig. 42), large ches—short drifts, in fact—are cut in the side-walls. The hitches are smoothed off short pieces of plank are laid cross-wise for the reception of a 12 by 12-inch bearer, feet long under each end-plate. Long flat wedges are driven between the foot-rds and the bearers, wedging the latter up against the end-plates of the set above. en everything is wedged tight, the drifts or hitches are carefully blocked up and set is lagged.

The method of cutting the stations and pockets is illustrated in Fig. 43.

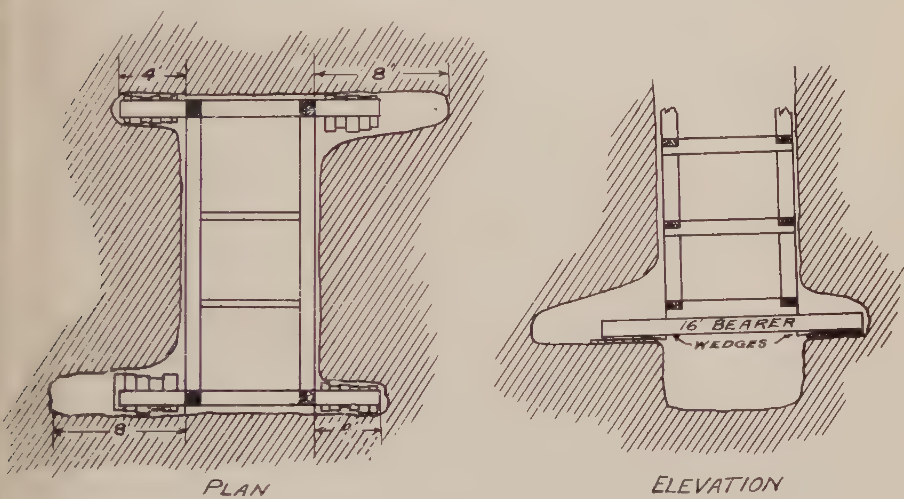
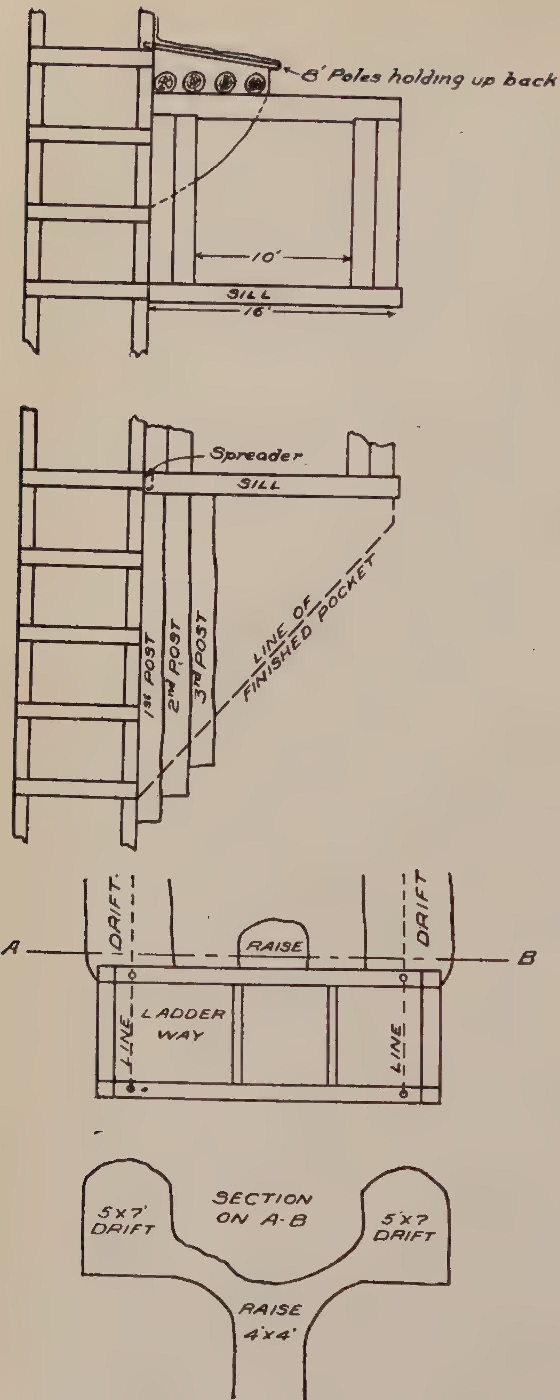


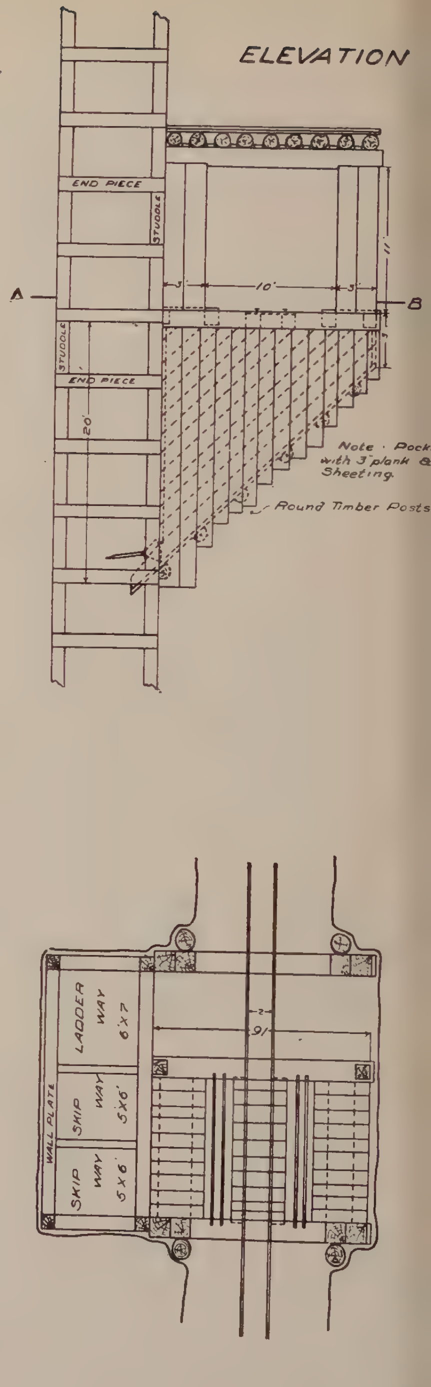
FIG. 42. Bearing Timbers.

Lines for two small drifts are set on the wall-plates 12 inches inside the studdles. ve 5 by 7 foot drifts are started on these lines, and, simultaneously, at a point 20 t below these drifts, a small raise is started. This raise runs up along the shaft, blocking and wedges encountered are removed, and the lagging is nailed to the ll-plate.

The drifts are stopped at 16 feet. The raise is holed through and is used as a ute while the station and pocket are being cut. Sills are laid along the length of e drifts, tops flush with the wall-plates against which the ends abut. The backs of e drifts are then taken down to a total height of about 15 feet. The station caps e heavy, 24- to 30-inch round timbers, 16 feet long, framed to 10 feet between the oulders.



Cutting Station and Pocket.



Completed Station and Pocket.

FIG 43.

The cap at each end rests on a double 18-inch post, 11 feet long. All timbers are firmly wedged in place.

To hold up the back while the ground between the sets is excavated, a cut is first made some 2 feet higher than the station set. The back is held by 8-foot poles, as shown in the illustration. When sufficient ground is cut out, 20-foot round poles 12 to 15 inches in diameter are laid across the caps. A small drift is usually started at one side of the station to make room for handling the long timbers. The poles are placed as shown in the view of the finished station, held apart by small blocks. They are overlaid by two thicknesses of board with the joints broken; the back is then carefully blocked up. The remainder of the station is similarly taken out and timbered; short transverse poles are used temporarily to hold the back; these are later replaced by 20-foot poles. There are about ten of these poles to a station of this size.

The lower section of the station is next taken out and, when the station is completely excavated, a third sill and set is placed between the outer sets at the ladder-way divider.

Finally the pocket is cut down beside the shaft. Posts are set under the sills against the studdles, extending 2 to 3 feet below the line of the finished pocket. As the pocket is cut back, other posts are placed against the first ones and tightly wedged in place. The station set sills are held apart at each end by a 12-inch square spreader or cross-sill. Four cross-sills of 16-inch square timbers are mortised into the outer sills of the outer sets. The two wall-plates in front of the station are cut off against the studdles, raised up even with the caps, and fastened. The pocket is lined with 3-inch plank covered with a wearing plate of ⅜-inch sheet iron. Quarter pan gates are used at the mouth.

Conditions on the Missabe make for slow progress. An average of 10 feet a week from start to finish on a 250- to 350-foot shaft, offering no special difficulties, would be considered fair. In the West, 20 to 25 feet a week through hard rock is not at all unusual.

The cost of sinking a typical Missabe shaft ranges from $50 to $100 per foot. This figure contemplates a shaft from 200 to 350 feet deep, a certain amount of quicksand in thin layers, but no special difficulties. The volume of water to be handled is a variable factor and the maximum figure allows for a large volume. The two shafts in the following table may be considered as typical of the Hibbing and Chisholm districts.

		Cost per foot			Supplies analyzed per foot			
	Depth	Labor	Supplies	Total	Explosives	Timber	Steel-sets	Misc.
A	213	29.99	24.61	54.60	0.49	10.00	.....	14.12
B	253	46.03	56.74	82.60	3.32	14.00	15.44	19.15

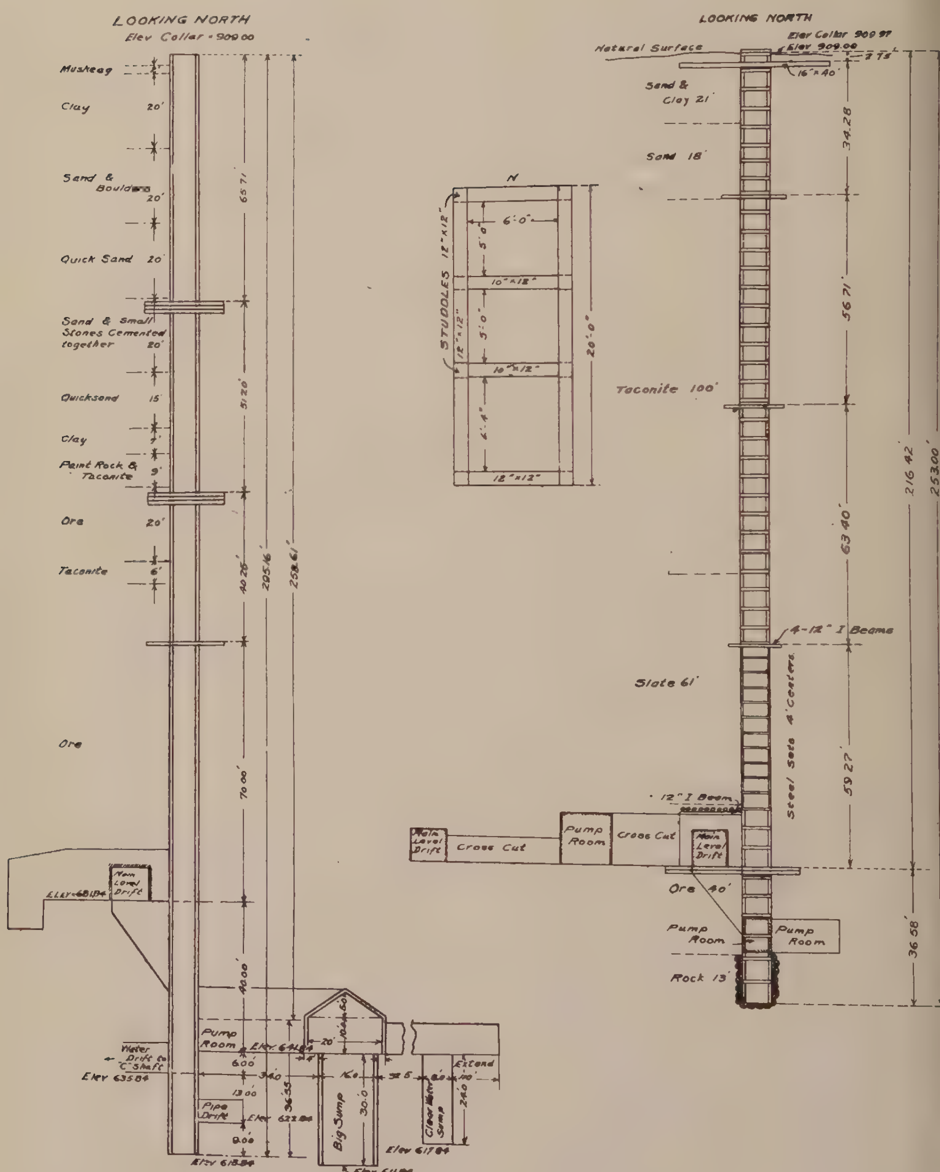


FIG. 44. Typical Missabe Shafts.

WEBB NUMBER THREE SHAFT

Month	No. Ft.	Boiler House Expense				Pumping Cost Labor and Supplies		Shaft Labor							Shaft Supplies					Steel Sets		Total Cost	
								Direct		Indirect		Total		Cost	Total Cost for the Month			Cost		Cost		For the Month	Per Ft.
		Labor	Fuel	Total	Per Ft.	Total	Per Ft.	Days	Amount	Days	Amount	Days	Amount	Per Ft.	Explosives	Candles	Miscel.	Total	Per Ft.	Total	Per Ft.		
Aug.	47	$74.36	$55.72	$130.08	$2.767	$218.04	$4.639	156.25	$368.52	54.6	$147.53	210.85	$516.05	$10.98	$9.35	$3.08	$557.85	$570.28	$12.13	$692.50	$14.73	$2,126.95	$45.25
Sept.	43	67.50	64.71	132.21	3.077	143.29	3.332	381.4	1,013.03	121.3	303.09	502.7	1,316.12	30.60	187.04	12.32	665.35	864.71	20.11	633.56	14.73	3,089.89	71.86
Oct.	65	69.74	145.08	214.82	3.305	249.93	3.845	425.2	1,208.69	163.4	425.96	588.6	1,634.65	25.16	196.71	9.24	565.49	771.44	11.87	957.71	14.73	3,828.55	58.90
Nov.	20	133.23	208.42	341.65	17.082	327.69	16.384	340.8	858.23	338.1	945.58	678.9	1,803.81	90.19	188.40	15.30	632.91	836.61	41.83	294.68	14.73	3,604.44	180.22
Dec.	37.5	186.82	313.05	499.87	13.329	405.62	10.816	500.4	1,243.14	217.8	544.49	718.2	1,787.63	47.67	146.66	9.17	693.37	849.20	22.65	552.53	14.73	4,094.85	109.19
Jan.	15.5	189.75	1,283.75	1,473.50	95.064	650.83	41.987	341.9	928.91	172.8	483.05	514.7	1,411.96	91.09	176.82	14.10	446.73	637.65	41.14	228.38	14.73	4,402.32	284.02
Feb.	20	233.20	2,447.87	2,681.07	134.053	740.11	37.005	524.7	1,359.64	403.5	1,168.80	928.2	2,528.44	126.42	108.10	12.90	412.70	533.70	26.68	294.68	14.73	6,778.00	338.90
June	19	141.34	1,736.52	1,877.86	98.834	587.68	30.930	892.0	2,340.80	501.3	1,353.61	1,393.3	3,694.41	194.43	79.80	16.75	784.18	880.73	46.35	280.11	14.73	7,320.79	385.30
July	4	52.39	237.90	290.29	72.572	110.04	27.510	160.2	416.32	57.9	174.52	218.1	590.84	147.72	41.15	8.13	104.12	153.40	38.35	58.94	14.73	1,203.51	300.88
Total	271	$1,148.33	$6,493.02	$7,641.35	$28.197	$3,433.23	$12.668	3,722.85	$9,737.28	2,030.7	$5,546.63	5,753.55	$15,283.91	$56.39	$1,134.03	$100.99	$4,862.70	$6,097.72	$22.50	$3,993.09	$14.73	$36,449.30	$134.49

COST OF POCKET AND STATION—NOT INCLUDED IN ABOVE FIGURES $8564.32

NOTE—The "Pumping Cost" includes all pumping expense outside of boiler house:
Cost of Fuel—$3.95 per ton.
Cost of Labor—Common, $2.10; Miners, $2.75 to $3.25.
Labor conditions: Fair—until water was encountered, then good men became very scarce.

PROGRESS—On the whole very satisfactory. Water never got the best of the men for more than a few hours. No drownings out of shaft.

HISTORY—

August, 1910—Sinking began August 14, 1910. 40 ft. surface, consisting clay, sand, boulders and hard pan; 7 ft. heavy broken taconite—all machine drilling—two ten-hour shifts.

September, 1910—43 ft. heavy broken taconite—all machine drilling—two ten-hour shifts.

October 1910—65 ft. progress—conditions unchanged until close of month when water was encountered.

November, 1910—20 ft. rock—conditions unchanged. Water grew to 500 gallons per minute and much time was taken to prepare for larger volume soon to be encountered, consequently there were about twenty shifts in which no sinking was attempted.

December, 1910—37½ ft. From 195 to 205 ft. a few thin seams of paint rock were encountered. These gave a good deal more water and at 200 ft. the water was 800 or 900 gallons per minute, requiring two Prescott sinkers at all times. There were three sinkers in the shaft and a No. 10 Cameron.

January, 1911—15½ ft. progress. No sinking was attempted until January 9th, on account of more preparations for handling water, because a 20 ft. layer of ore and broken taconite had been reached. It was known that the water would greatly increase in this layer, which it did up to about 1,500 gallons per minute. Three Prescott sinkers 14x8x12 were in continuous operation during January and afterwards.

February, 1911—Solid taconite was encountered at 233 ft. on February 3rd, and three eight-hour shifts were started on the 16th. A depth of 248 ft. was reached by March 1st and sinking was stopped to cut station and do considerable drifting. Prescott compound pumps 9x16x10x18 were then put in place on the bottom level, and from June to July 5th, inclusive, solid rock sinking went on to completion.

"A" Shaft. Sunk in the ore-body under normal conditions; very little water; depth a little under the average.

"B" Shaft. Sunk entirely in rock, 60 feet being very hard sinking through seamed taconite. For 45 feet above station this shaft is supported by steel sets. The cost of support is given in timber cost per foot for timbered section and steel cost per foot for steel-set section.

On the accompanying insert will be found details of the Webb shaft, furnished through the courtesy of the management. It is to be noted that the cost of sinking through 47 feet of ordinary drift is \$45.25 per foot. Of this, \$4.50 is chargeable to pumping, \$14.73 to steel for support, and \$12 to supplies. Over 100 feet sunk through hard taconite requiring machine work, cost on an average \$65 a foot, including \$4.50, \$14.73, and \$16 per foot respectively for pumping, steel framing, and supplies. The succeeding months show a heavy increase due to influx of water.

In Fig. 44 are shown two typical Missabe shafts with legend of ground passed through. The left-hand shaft was sunk in the ore-body and the other one in the rock. About 40 feet of this shaft is lined with steel sets.

As early as 1900 the Oliver Iron Mining Company began using steel for lining mine shafts. In Volume VIII of the Proceedings of the Lake Superior Mining Institute, Frank Drake describes in detail the construction of steel sets at Pioneer "B" shaft, Ely, Minnesota. The discussion includes an interesting comparison of comparative costs for steel and wooden framing. Fig. 45 is copied from this article. The shaft is a 70° incline, 6 by 17½-foot inside dimensions, having two 6 by 6-foot skip compartments and a pipe and ladder compartment. The wall-plates are 30-lb. rail, end-plates 25-lb. rail. It is stated that only when conditions require 50-lb. rail would I-beams be more economical. The members of the set are riveted together with 3½ by 3½ by ½-inch angles secured by ½-inch rivets. Studdles, 4 feet long, are bolted so that they can not be knocked out. Hangers are made from ¾ by 2-inch iron bars. The lower end is formed into a hook and the upper end bent at right angles to hold a screw with vertical motion.

Wooden lagging of 2-inch plank is used on account of the greater cost of the metal substitutes—either corrugated, galvanized $\frac{1}{16}$-inch steel or buckled plates. The metal lagging has the great advantage of being fire-proof. An all-steel shaft is fire proof—a steel set with wooden lagging is not, and one of the great advantages of the steel set is thus lost. This would seem to be a sacrifice of a very important advantage for a small economy. To minimize this objection metal lath was used for 16 feet covering 4 adjacent sets dividing the shaft into 100-foot sections with 4 adjacent sets completely fire-proofed.

The relative strength of 2-inch plank, $\frac{1}{16}$ inch galvanized, corrugated steel and buckled plates is respectively 360, 200, and 400 pounds per square foot distributed load. The weakness of the corrugated plate is apparent. In heavy ground this could of course be remedied by using two thicknesses. The shop work is simple. At the normal rate of sinking the regular shop crew can easily make all the parts for the lining of the shaft in addition to the regular work. The steel is purchased cut to the

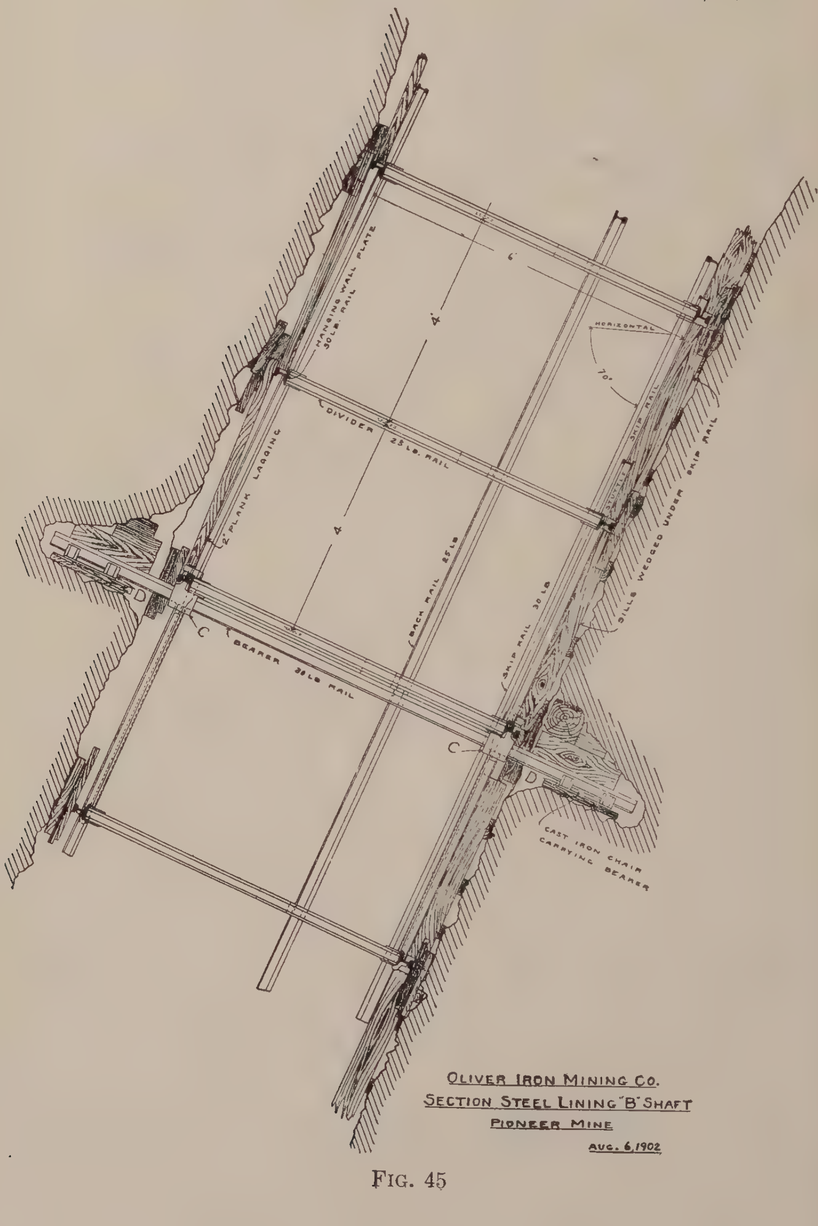

HANGING WALL PLATE 30 LB. RAIL
6'
4'
4'
HORIZONTAL
70°
SKIP RAIL
DIVIDER 25 LB. RAIL
2" PLANK LAGGING
STUDDLE
SILLS WEDGED UNDER SKIP RAIL
BACK RAIL 25 LB
SKIP RAIL 30 LB
BEARER 30 LB RAIL
C
C
D
D
CAST IRON CHAIR CARRYING BEARER
OLIVER IRON MINING CO.
SECTION STEEL LINING "B" SHAFT
PIONEER MINE
AUG. 6, 1902

FIG. 45

uired lengths. Templates are made for each member and the rivet holes are lled with an ordinary drill-press. The studdles are slotted by a power-operated Hangers are easily made. Sets are assembled on the surface, lowered in a skip mpartment on a special truck; as the compartment is 6 by 6 feet the set goes down h the end-plate lying diagonally across the compartment. After it has passed the t set in place, it is swung in position and hung from the latter. Since the wall-plates 18 feet long, the shaft bottom must always be at least 14 to 15 feet from the last before another wall-plate can be swung in place.

Wooden blocks and wedges are used to line up and secure the shaft. Wedging is fined to points of connection between wall-plates and end-plates or dividers. When sufficient number of sets are in place, the skip-rails and back- or guard-rails are l. After a new rail is suspended from the last rail in place and carefully lined, t holes are drilled with a pneumatic drill through the flange of the rail and of the ll plate in the same operation. Bearers of 30-lb. rail are put in at suitable distances ording to the rock. These bearers are supplied with two chairs, *C* in Fig. 45 for the per support of the steel shaft-set above and of the studdles of the next set below. e bearers rest on chairs *D* in ample hitches cut in the walls. Wooden wedges are ven under the base of these chairs until the bearers are wedged as tight as possible the last shaft-set. The space over the chairs is then blocked. These hitches may from 2 to 4 feet long. Bearers range from 50 to 100 feet apart. After the arers are in the shaft, the hangers are removed and used again.

The cost of materials for steel lining per foot of shaft varies according to the aterial used for lagging as shown by the following table:

	Cost per foot of shaft		
	Wooden lagging	Wood and steel lagging	Steel lagging
(1) Cost of steel sets	$5.79	$5.79	$5.79
(2) Bearers and chains	.41	.41	.41
(3) Hangers	.23	.23	.23
(4) Lagging	2.12	2.73	5.94
Total	$8.55	$9.16	$12.37

The cost of materials for wooden set per foot of shaft at the average price of 15.50 per thousand for shaft timbers and lagging would be $5.73 as shown by the llowing summary:

(1)	Cost wooden sets	$3.10
(2)	Bearers	.17
(3)	Hangers	.11
(4)	Lagging (5-foot centers)	2.35
		$5.73

The wooden materials for lining, therefore, cost $3.43 less (37.5 per cent) than l-steel lining. It must be remembered that these figures do not include skip, rail aterial, wedges, blocks, nor the shaft labor in placing the sets, which items are

considered to balance. The saving in excavation should offset this increase to som extent. It would seem safe to assume a saving of 10 per cent in the excavation cos

The following table gives the comparative costs of sinking the Pioneer "B" an No. 2 Savoy Shafts. These shafts are a few thousand feet apart and were sunk sin ultaneously by the same company. The Pioneer Shaft was contracted in so far as ad vance at the face is concerned.

COST OF SINKING "B" SHAFT, PIONEER MINE, AND NO. 2 SHAFT, SAVOY MIN

	"B" Shaft Pioneer Mine	No. 2 Shaft Savoy Mine	Excess cost of Pioneer Shaft	Excess c of Savo Shaft
Dip	70°	83° 30′		
Material of sets	Steel	Wood		
Material of lagging	Partly wire rope and partly wooden lath	Wooden lath		
Size of timber sets		All 10″x12″		
Compartments	2-6′x6′ 1-5′x6′	2-6′x6′ 1-4′8″x6′		
Outside dimensions	6′6″x18′	7′8″x20′		
Outside area	117 sq. ft.	153 sq. ft.		
Progress per working day	1.43 ft.	1.54 ft.		
(1) Contract price of shaft (based on price paid contractors)	$15.95		$0.91 (for items 1 and 2)	
(2) Cost for company-account labor employed in sinking, cutting hitches, cutting, sinking pump stations, etc., and including all other company-account labor not properly included under (3), (4), (5), and (9) below	5.74	$22.60		
(3) Labor expended in laying or attaching skip rails, back rails, and parts of wooden sets, but not including any miners or other labor included under (1) or (2) above	2.03	1.45	$0.58	
(4) Miscellaneous labor regularly employed (engineers, firemen, landers, pumpmen, etc.)	10.71	8.19	2.52	
(5) Shop and team labor	1.74	2.03		.29
(6) Material and supplies:				
(a) Explosives	1.86	1.98		.1
(b) Timber and lath for sets		5.26		5.2
(c) Mining timber	1.87	.63	1.24	
(d) Iron and steel	1.52	1.15	.37	
(e) Pipe and fittings	1.17	.88	.29	
(f) Steel rail	4.70	1.10	3.60*	
(g) Wire rope for lining	.37		.37	
(h) Miscellaneous supplies	4.68	2.81	1.87	
(7) Fuel	4.30	2.93	1.37	
(8) Air	2.89	1.33	1.56	
(9) Temporary surface work (buildings, head-frame, etc.)	.28	.58		.3
Total cost per foot	$59.81	$52.92		

*Includes material in wall- and end-plates

In Volume X of the Transactions of the L. S. M. Inst. appears an article T. R. Thompson describing a much lighter construction used a couple of years la on one of the lower ranges, viz., a three-compartment inclined shaft, consisting two hoisting compartments 7 feet 3 inches by 5 feet 3¼ inches, and a third co partment about 9 inches narrower.

Each compartment is a complete rectangle of 4 by 3 by $5/16$ in angle iron, the ree compartments being riveted together, the divider being in this way made stronger. e sets are held together by separators made of ¾-inch round bolts extending through -inch gas pipe, the latter acting as studdle, the former as hanger. Bearers at 100-ot intervals.

Cost	Total	Manufacture		Installation	
		Labor	Materials	Labor	Materials
er set	$53.02	$6.06	$19.63	$24.01	$3.32
er foot	8.83	1.01	3.27	4.00	.55

The weight of a set is 1,060 pounds; the cost per pound is 2½ cents; cost alled, per pound, 5.1 cents. The cost of rail in place is $8.65 per set, or $1.44 r foot, so that the shaft set with skip rails and guard rails costs altogether $61.67 $10.278 per foot.

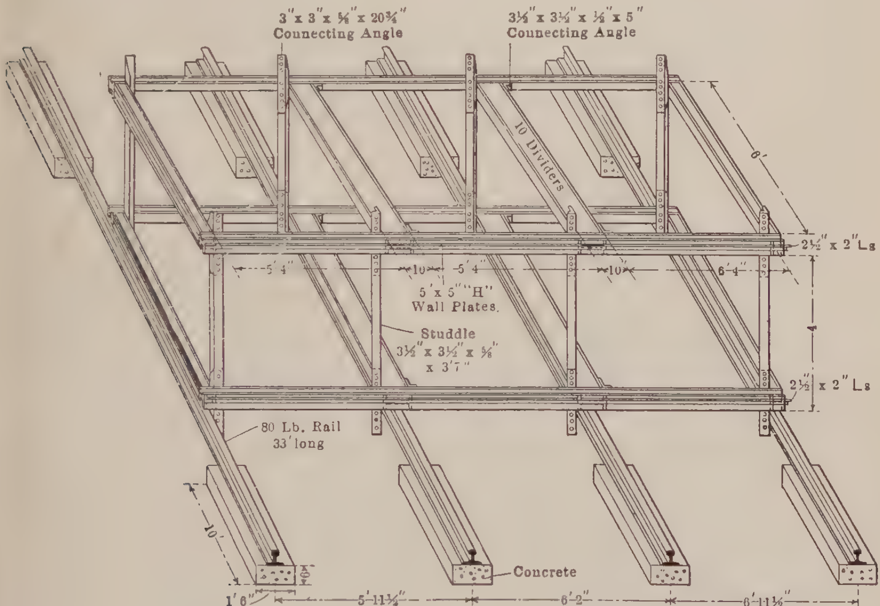


FIG. 46. Steel Sets used at Webb Shaft.

Fig. 46 shows the steel sets used at the Webb Mine (see insert). The inside easurements are 6 by 18¾ feet. The wall-plates and end-pieces are 5-inch H's eighing 18.7 pounds per foot. The dividers are 10-inch I-beams. The sets are paced 4 feet center to center and held together with eight 3½-inch angle studdles; all ngles are shop-riveted to the end pieces and dividers so that in a wet shaft less than wo hours' time is required to put a set in place.

The shaft is lagged with Norway pine; the lagging is held in place by small 2 by ½ by 2-inch angles shop-riveted to the back of all wall-plates and end-pieces. Later on it is intended to replace the lagging by concrete. The drawing shows the first shaft bearers placed under the second set. The rails are 33 feet long, resting on concrete foundations each 10 feet long. In the shaft 12-inch I-beams 14 feet long are used for bearers at 60-foot intervals. The cost per foot of shaft for steel support is $14.73.

SINKING UNDER DIFFICULTIES

In 1903 a shaft was sunk at Hibbing through 57 feet of dry sand followed by 50 feet of quicksand and 31 feet of hardpan. Because the management anticipated much trouble in holding their shaft through the quicksand a small experimental shaft to serve later as pump and timber shaft was decided on. The experience gained from this would be a help in planning to meet the greater difficulties sure to appear in excavating a much larger area. When shafts are sunk through quicksand, the trouble increases in much greater ratio than the relative areas of the shafts bear to each other. The work of sinking the large shaft would also have the advantage of drainage effected through the smaller one.

The timber shaft was started 8 by 10 inside timber measurement. The support was 12 by 12-inch timber sets with 3-foot studdles, 1¼-inch hanging bolts, using 24-inch round bearers, 32 feet long. At 45 feet the lower set was supported by ¾-inch wire ropes passed through the bearers on the surface, using ½-inch wrought iron plates, 8 by 8 inches being used for washers. When quicksand was struck at 57 feet the pull on the wire rope increased markedly and the bearers were reinforced by the building of two trusses, each bearer being used as the lower chord of a Queen Post truss.

At this point it was decided to change from the suspended set method and try a drop shaft. A shaft set was beveled to an edge, placed in position under the bottom set, and forced down with jack screws. As this set penetrated the ground, additional timbers were slipped under the jack screws, each one being bolted successively to the timber below it. In this way a depth of 85 feet was attained, hoisting the sand and pumping the water. At this point the difficulty became so great and the progress so slow that the size of the excavation was reduced to 6 by 8 feet, a new shoe of that size was forced down and added to as it penetrated the quicksand. Meanwhile the bottom set at the 85-foot level was anchored to the trussed bearers at the shaft collar by four 1¼-inch wire ropes.

By slow and careful work the inner crib was forced down 22 feet into hardpan at a depth of 107 feet. Here again the bottom set was anchored by wire ropes to the collar set. The strain on these various sets of ropes, which took the place of shaft bearers, was evidenced by the fact that the 8 by 8-inch washers were pulled from 2 to 3 inches into the timbers. The actual advance through quicksand was made by forcing down ahead of the timbers in the center of the shaft a box made of 1¼-inch sheet steel 3 by 5 feet in area and 10 feet long. This helped to drain the bottom, solidifying the sand, acting as a sump for the suction pump. By constantly repeating

the following succession of operations—pumping out the water slowly, hoisting out the sand, jacking down the box, and finally the surrounding timber structure—the shaft was advanced a few inches a day, making from 8 to 10 feet advance a month. The pumps handled from 300 to 400 gallons of water a minute.

The great danger in sinking through quicksand is the temptation to move too fast, causing the sand and water to bubble under the bottom timbers. The hydrostatic pressure in the water-bearing sands surrounding the shaft increases as depth is gained and quickly becomes a dangerous force that must be very carefully handled. Sand-boils in the bottom must be kept down by driving or forcing timber, sinking

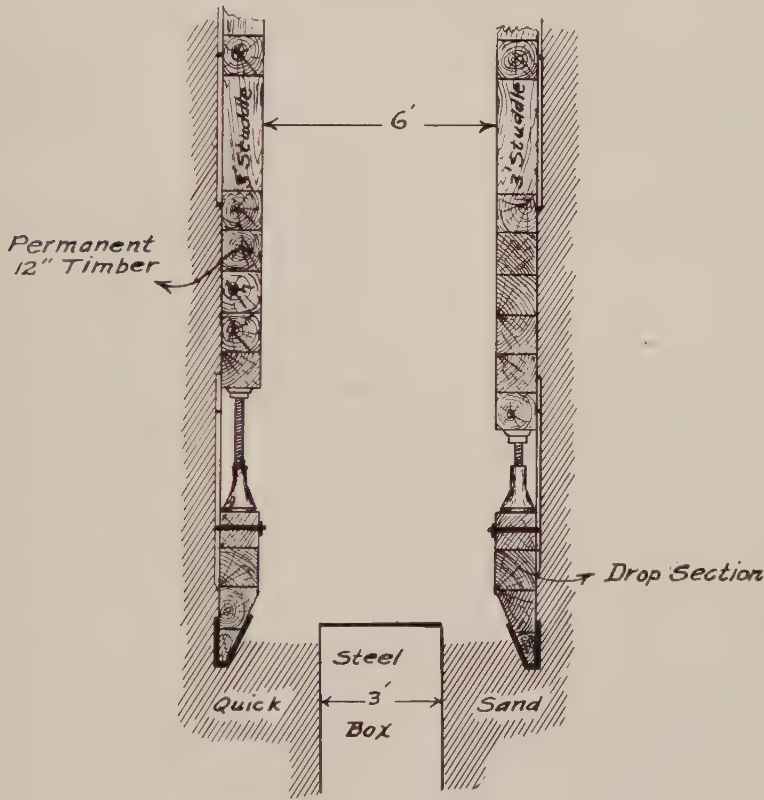


FIG. 47

a drainage well, if possible, to separate the water from the sand, beating the shaft down by seemingly insignificant gains. Continuous, slow, cautious work will count in the end and is the only thing that will keep the shoe level and prevent the shaft from tilting and perhaps shearing. The problem is slowly to drain out and pump up the water, at the same time to hoist the minimum of sand. If the latter is removed too fast from within the shaft, the sand on the outside will run in and the shaft may be lost in a few minutes. When a serious sand-boil occurs, it is customary to let the

shaft fill several feet, 15 to 25 if necessary. It is better to do that and slowly recover it than to run the risk of wrecking the timbers and losing the shaft.

In the hardpan progress was made by driving down lagging, or lath as it is locally called, excavating the ground, and placing the timbers, bolting them together. Progress in hardpan and clay averaged 1½ feet per day and the shaft reached the ore at 138 feet only 2 feet out of plumb. The total time required was seven months. This may seem very slow progress; the fact is, however, that one familiar with conditions would undoubtedly consider it a very satisfactory piece of work.

The hoisting shaft was commenced some months later, 40 feet distant; the size was 6 by 16 feet with standard sets to the water line. The drainage effected by the timber shaft made it possible to handle all the water in the new shaft with a No. 7 Cameron. When the quicksand level was reached, the shaft timbers above were well braced and the bottom set was hung by wire cable from the collar set. A different method was used to penetrate the quicksand, similar to the method used at the Ladd Shafts in Illinois, fully described and illustrated in the School of Mines Quarterly of Columbia University, Vol. 16. A drop section was built as shown in Fig. 47, made of four rounds of 12 by 12 timbers bolted together. The lower two sets were beveled flush outside and shod with ⅛-inch steel plate, the bearing edge of the cutting surface being about 3 inches wide. A plank sheeting is bolted to the shoe, extending about 4 feet above the shoe proper. Between the shoe and the bottom set of the regular shaft timbers some thirty 2½ by 18-inch jacks were placed. The jacks forced down the shoe, the outside sand being restrained by the overlapping sheeting. As the shoe penetrated the ground, the jacks would be lowered on one side so that it was possible for a timber to be added to the cribs overhead, the shaft being held by the jacks under the remaining three sides. The timber crib would thus be added to one side at a time. The greatest care was necessary to prevent loss of alignment.

This process was continued with an average progress of 9 feet per month (working three 8-hour shifts daily except Sunday), to a depth of 107 feet, when the shoe penetrated the clay or hardpan. The time consumed in sinking 107 feet was about 130 working days. As in the timber shaft, further progress in the hardpan was made by driving lath; this made possible the removal of the ground and the putting in of timbers. Progress in this part of the work averaged 18 inches daily.

At the Bangor Mine a timber shaft is being sunk under similar conditions. In 12 days the shaft was sunk and timbered 51.5 feet to water. It was planned to complete the shaft by raising when the nearest mine heading, then 300 feet away, should be extended under the timber shaft. This caused a 3 months' delay in sinking. Meanwhile a churn-drill hole was put down and cased to aid the draining of the shaft. The raise was duly started, but after passing through 4 feet of mixed sand and boulders the men were driven out by quicksand and it became apparent that the raise must be abandoned. The shaft was sunk a few feet deeper to 59 feet. The sand proved to be very wet and it was very difficult to separate out the water and drain it below the place for the next timber set, yet this was necessary.

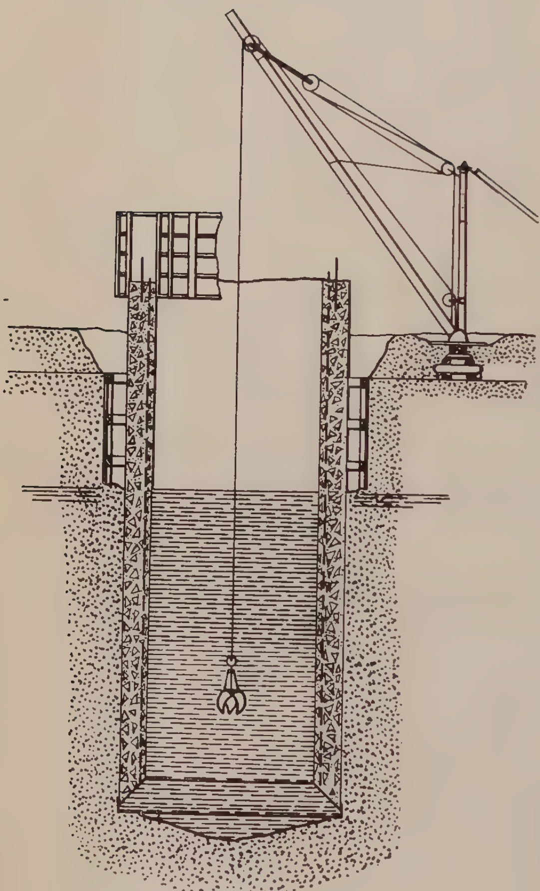
The method used was to jack down a 36-inch steel sinking cylinder 6 feet long, surrounding the 6-inch casing pipe in the center of the shaft. When the top of the box is flush with the water in the shaft, the sand within the box is dug out to a depth of 4½ to 5 feet, giving a 5-foot sump or well. The central drill casing is cut down so that the top of the casing is 2 feet below the top of the cylinder. As the water in the surrounding sand slowly drains out and seeps under the bottom of the cylinder, it raises around the casing pipe and finally drains off through it. (In a straight sinking job with no underground workings to drain into, the well would be used as a sump for a sinking-pump suction.)

The sand drains very slowly and the process is hastened by excavating into the sides of the shaft and making room for pieces of lath 2 by 10 and a little longer than the regular lagging. These are slipped one at a time behind the lagging and lagging strips of the set above and held in place by banking up sand against them on the inside of the shaft. The central well is covered over and the sand from the excavation is piled on top of this cover. When the whole perimeter of the shaft has been treated this way, the sand usually drains out sufficiently to permit the putting in of another set. The lagging strip holds the lath from crowding in and the lagging is subsequently slipped in in the usual way. The sand from the excavation for the timbers is usually dumped for the time being in a pile in the center of the shaft and removed when convenient. The process is repeated in cycles as follows:

1. Jack down the central cylinder or box.
2. Remove a section of casing pipe.
3. Cover the drain box and commence excavating for lath.
4. When the perimeter of the shaft is completely lathed and the water has sufficiently separated out from the sand, the excavation may be carried deep enough around the sides of the shaft to make room for a set of timbers.

Sometimes the sand drains out so slowly that several days and even weeks elapse between the two cycles. Such is the case at the Bangor timber shaft where progress is very slow. There is no way of hastening this process. The strain on the timbers is often excessive; they are reinforced by numerous short studdles or blocks. Instead of the usual bearers hitched into the walls, pieces of timber are placed across each corner of the shaft forming a seat for the end-plate and the wall-plate at each corner. These bearers are hung by ¾-inch to 1-inch wire ropes from stout surface trusses, made of heavy round timbers, well formed and strongly bolted together. As depth increases this is repeated. There may be several ropes in each of the four corners of the shaft, each rope supporting a bearer at a different horizon in the shaft.

Caissons.—The caisson or drop-shaft method of sinking, of which the foregoing may be considered a modification, is gaining in favor in the United States for the penetration of soft ground. Caissons are smooth-walled, hollow drums, circular or rectangular in shape, that sink of their own weight as the ground contained within them is removed either by the ordinary process of excavation or by some mechanical device such as the orange-peel or clam-shell dredge bucket, as shown in Fig. 48. For



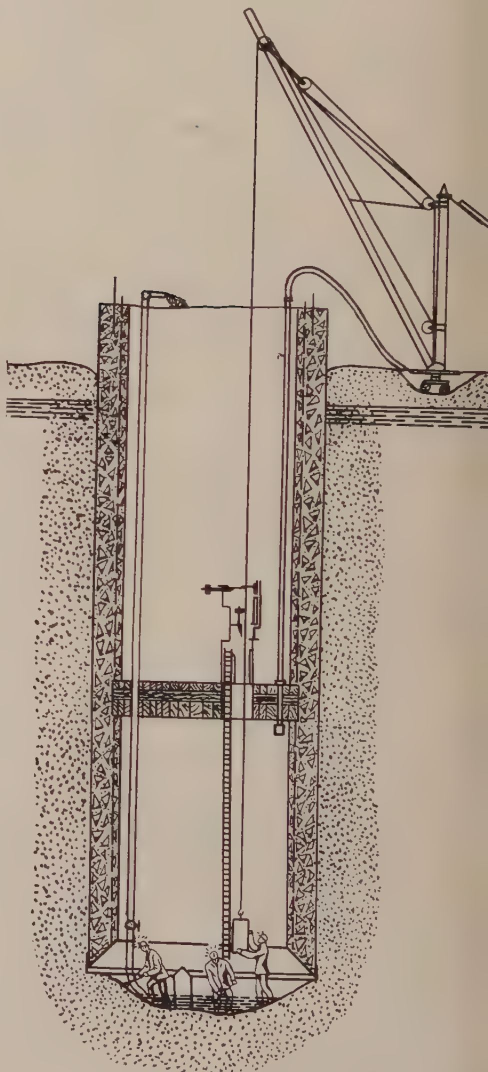


FIG. 48 Caissons. FIG. 49

small shafts the circular caisson is cheapest. For shafts of more than three compartments the rectangular caisson gives the least waste space though it entails greater construction expense. Modern caissons are made of steel or of reinforced concrete built up on a steel shoe. The thickness of concrete caissons runs from 24 to 60 inches on the bottom, tapering upward, the outside wall being flush. The caisson is added to from the top as the shoe penetrates the ground.

When the caisson reaches the solid rock, it must be "sealed" to the rock so as to make it sand- and water-tight. Occasionally this is accomplished by the shoe passing through an intermediate decomposed clayey stratum between the sand and hard rock. A caisson may be sealed in various ways. If mud is simply oozing through beneath the shoe, this may often be stopped by caulking under the shoe with wedges and rags. The mud will then dry and harden. Sometimes it is necessary to cut a more even bearing for the shoe and then systematically block the crack by driving small dry wedges until the water is held back. If the water is coming in through distinct feeders, it may be piped to a central point, the ledge on which the caisson rests may be trimmed and a concrete dam built on this rock wall to the caisson, surrounding the water pipes. Several small pipes are built into the dam. When the concrete has set sufficiently, the drain pipes may be plugged. Small leaks may be stopped by forcing grout through the small pipes into the space behind the dam.

A caisson may go down part of the way and encounter more water than pumps can handle. The remedy then is to sink the caisson under air pressure provided the solid ground is not over 100 feet below the water level, as shown in Fig. 49. The caisson is provided with a cover and an air-lock or equalizing chamber for the passage of the men and the dirt; compressed air is forced into the caisson until the internal air pressure overbalances the outside hydrostatic pressure. The result is a dry shaft bottom, the water being forced back into the surrounding sand. As the bottom is lowered and the caisson sinks, the increasing hydrostatic pressure must be met with a corresponding increase in the air pressure. Instead of draining the surrounding country and lowering the water level the water is simply held back by air under a pressure equal in pounds per square inch to a little less than one-half the head (in feet) of the surrounding water.

The limit of human endurance in the matter of air pressure regulates the depth attainable by this method. The men work on the shaft bottom in an atmosphere 3 to 4 times as dense as on the surface. For each 34 feet below the hydraulic level the air in the caisson is a little more than doubled in density. It is generally stated that the economic limit under which this work may be carried on is 4 atmospheres, roughly 60 pounds of air pressure, which would theoretically set the limit for sinking under air at about 100 feet below the water level. The progress of the caisson is somewhat retarded by the air pressure in the sinking chamber, and, when the caisson is too light for the depth to which it must sink, it sometimes becomes necessary periodically to greatly diminish the pressure to sink the caisson. Frequently the weight of the caisson is insufficient to overcome skin friction and air pressure. Heavy, con-

veniently-shaped cast-iron weights are used to help sink the caisson; the upward pressure to be overcome may be 500 tons or more.

Some engineers and contractors use two air-locks, a man-lock and an excavation-lock. Others simply use one lock, dividing the shaft below the lock into two compartments to prevent the men being struck by the passing buckets. Whether one or two locks be used, the principle is the same and a description of the Moran air-lock will suffice. The passage through the lock takes place in three stages as illustrated in Fig. 50.

The lock is fitted to the temporary cover or deck. The joint must be air-tight. There are two horizontal doors hung so they may be swung vertically downward. The upper door closes the outer (atmospheric) entrance to the lock; the lower one separates the upper chamber of the lock from the working chamber which is open to the shaft. A pipe connects these two chambers. This pipe is fitted with a three-way cock

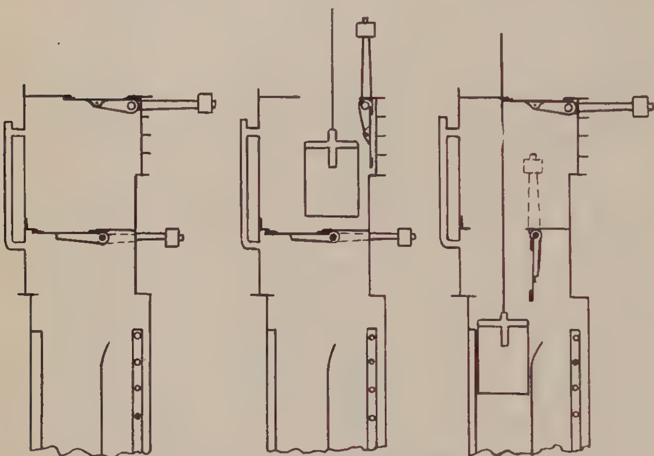


FIG. 50. Air Locks.

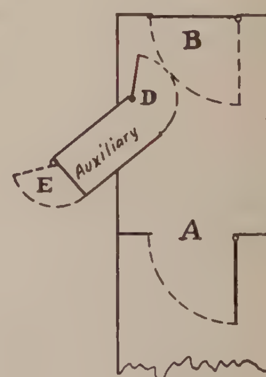


FIG. 51

by which either the upper or lower chamber is put in communication or the upper chamber may be opened to the atmosphere. These doors are hung and counter-weighted so that they are held closed by the excess air pressure beneath. When the pressures on both sides of the door are equalized, the door may be opened. In the middle view in Fig. 50 this has taken place and the empty bucket has passed through. Pulling the bucket over permits the shutting of the door. By the turning of a three-way cock so as to exclude the outside air the upper and lower chambers are connected, compressed air rushes in, and the pressures are quickly equalized. The lower door is now opened and the bucket descends. The reverse of this operation is necessary to hoist the bucket. The Moran lock is about 5 feet in diameter and 8 feet long, with a ladderway cylinder of smaller diameter and varying length.

Fig. 51 illustrates a modification used by some contractors containing an auxiliary air-lock at the side of the main lock. The main air-lock is closed to the atmosphere by

the door *A*. The door *B* is opened to the shaft. The auxiliary *A* has two doors, *C* opening inward and *D* outward. In the illustration *C* is open. The bucket discharges into the auxiliary lock until it is full. Then *C* is closed, *D* is opened, and the muck discharges into the main shaft. The receptacle used and method of hoisting is a detail subject to variation.

The miners are locked through in the same way with this exception that more time is consumed. Too rapid compression causes terrific pains in the head and ears, known as "blocking of the ears." Too rapid decompression causes much more serious trouble, a form of paralysis common to caisson workers, known as the "bends." Men accustomed to the work can lock through in from 2 to 5 minutes for pressures ranging from 20 to 25 pounds and can work from 8 to 6 hour shifts. When the pressure increases to 45 pounds, it takes 25 to 30 minutes to lock through and the shifts are reduced to 45 minutes, each man working 2 shifts.

The position of the air-lock varies. The safety of the men demands that it be a reasonable distance from the shaft bottom. A break in the pipes or caisson might occasion a sudden rush of water. A shaft fills very rapidly under such conditions and, if the quarters are too confined, the men may be drowned in the caisson chamber. A reasonable length of air-shaft with the air-lock on top would safeguard the men from this danger. The air pumped into the caisson by the compressor escapes under the cutting edge and bubbles up to the surface. This bubbling has some effect on the water surrounding the caisson, reducing the hydrostatic pressure to some extent.

When the ground passed through is filled with boulders, it is often necessary to excavate several feet below the cutting edge of the caisson in order to sink it. A "blow-pipe" is used to force out the water which collects in large pools in these depressions. This is a 4- to 6-inch pipe passing through the deck over the top of the caisson. The pipe has a stop-cock at the lower end to which a suction hose is connected. Above the stop-cock is another valve through which air may be admitted to the pipe.

When the stopcock is open, the water will be forced into the pipe by air pressure on the surface of the water. When the valve above the stopcock is opened, air will rush in, mix with the water and lessen its specific gravity. Under proper regulation, the weight of the water column can be made less than the air-pressure on the water surface in the shaft bottom. The result is that the water is forced to the surface. When the volume of water is heavy and the lift is great, the efficiency of the blow-pipe may be increased by substituting air under high pressure for the air admitted through the valve. Fine sand and silt can be blown out through the pipe and many caissons have been sunk on foundation work without hoisting any dirt. When rock is reached, the shattered condition of the first rock layers often makes it necessary to continue the excavation for several feet under air pressure. This may be done though it requires the most careful work under experienced supervision.

When a pneumatic caisson reaches the ledge, it is a simple matter to seal it to the ledge by cutting into the latter and building up a concrete seal. By laying a heavy strip of canvas over the crack, tacking it to both the inside of the caisson and the rock, the concrete may be laid without danger of its becoming porous. Without

this precaution the air pressure would blow the grout out of the concrete. The air must be kept on until the concrete has set. When the caisson is completed and sealed, the air lock is removed, the deck cut out, and all holes are patched. The shaft is then sunk in the regular way, bearing timbers are put in and wedged up to the caisson and the regular shaft lining carried up inside the caisson to divide the shaft into the desired number of compartments.

If a fire-proof shaft is desired, steel bearers are put in under the concrete section; heavy cast-iron brackers are drift-bolted into the walls, holes having been bored for this purpose. Horizontal sets to rest on these brackets are made of channel irons, corrugated iron lining to sheath the compartments, completes the job.

Woodbridge shaft.—In 1910 the Foundation Company sunk a shaft under air pressure at the Woodbridge Mine near Buhl. The site of the shaft is low and the drift consists of sand, gravel, and also contains much water and quicksand. Water level is at about 30 feet below the shaft collar. An ordinary timber shaft was started and soon abandoned for a concrete drop shaft to be sealed in the rock at a depth of 100 feet more or less. Contract price said to be $500 a foot.

The shaft is circular, 29 feet outside diameter with a rectangular opening 11 feet 10 inches by 16 feet 10 inches at the bottom. The opening is made a little larger than is necessary to allow for slight deviation from the perpendicular during sinking. An ordinary hoisting derrick was rigged beside the shaft site to handle the muck and also the concrete. The sinking plant consisted of a battery of three 60 h. p. boilers, a 7 by 10-inch Lidgerwood hoisting engine, and two air compressors.

The force consisted of a superintendent and 45 men divided as follows:

1 Air foreman.
1 Outside boss.
1 Timekeeper.
24 Shaft-men (8 to 10 per shift).
4 Hoisting engineers (2 on each 12-hour shift).
4 Air lock tenders (2 on each 12-hour shift).
2 Compressor engineers (12-hour shifts).
2 Firemen (12-hour shifts).
1 Blacksmith.
1 Blacksmith helper.
1 Teamster.
4 Laborers.

The skilled labor is generally hired in the East and moved from one job to another. Wages range upwards from $3 per 8-hour shift for shaftmen.

Sinking details.—A 14-foot hole was dug, the muck being hoisted with the derrick. The shoe was then assembled. Dimensions, outside diameter 29 feet, height 10 feet. The outside plate is vertical, while the inside one sloped upward from a 2-inch cutting edge, forming a V. The plates are connected by channels and angles at the V, a combination of plates and angles being used higher up. Reinforcing rods were tied to the shoe and concrete poured in. The sides of the shaft were built up in 5-foot

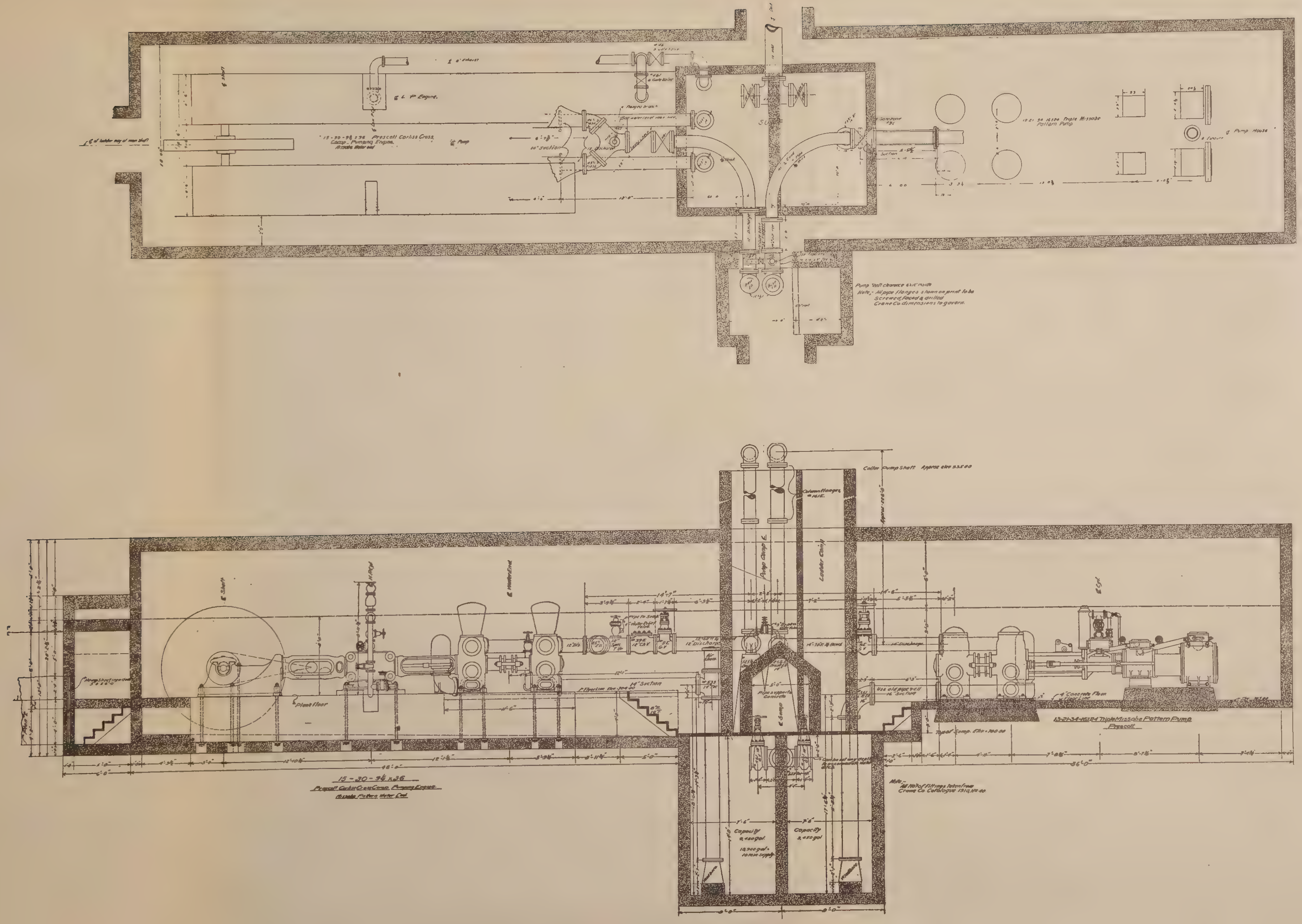


FIG. 51.—Pump Station and Pump Equipment

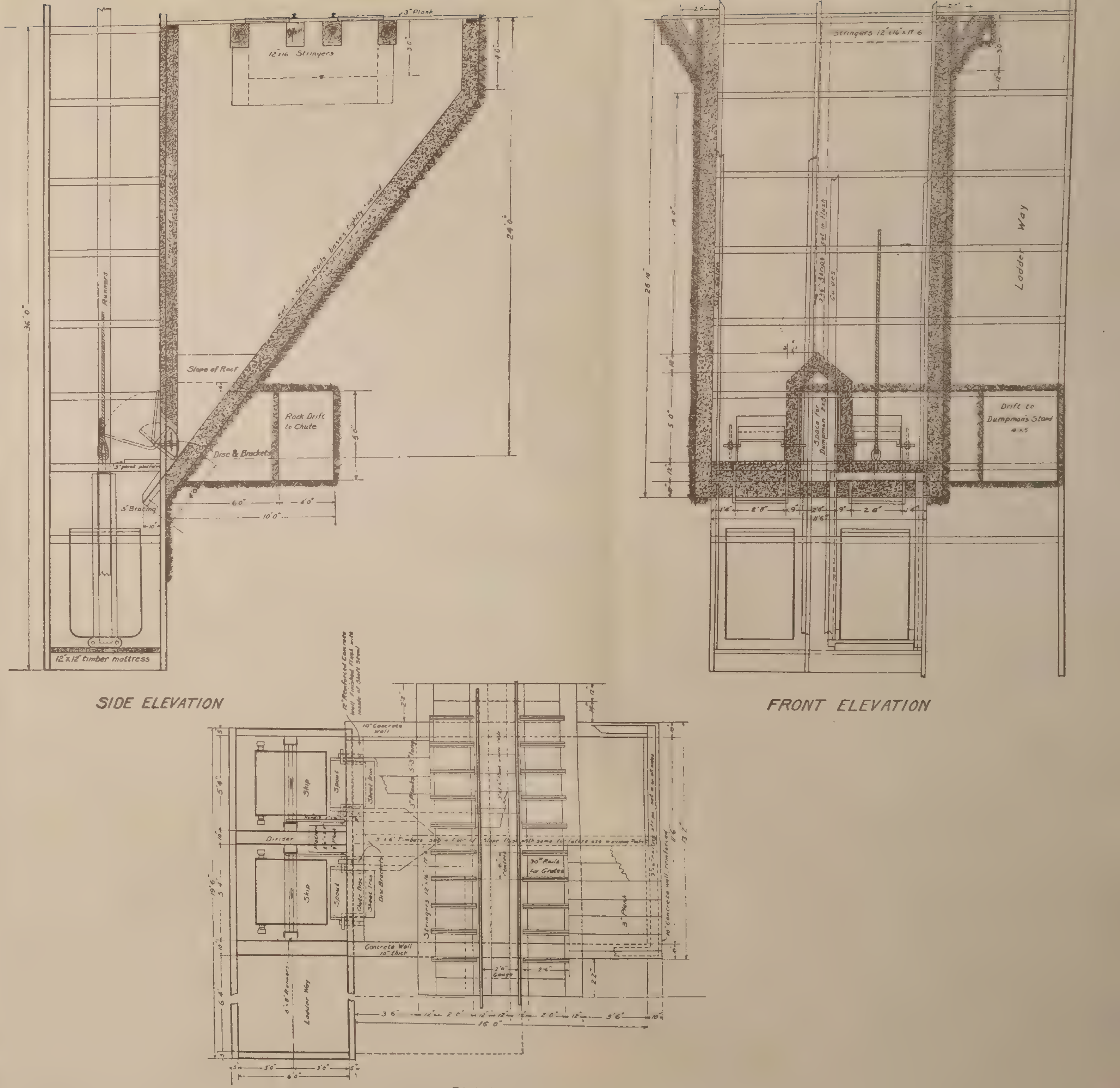


FIG. 52.—Shaft Pocket and Loading Station

sections, through the employment of a steel outer form made in 6 sections and a wooden inner form with steel rod reinforcement. The outer sections are 5 by 15 feet or $\frac{5}{16}$-inch steel plates; $3\frac{1}{2}$ by $3\frac{1}{2}$ by $\frac{5}{16}$ angles riveted into the sides and ends. These sections are bolted together to make the outer form. From 2 to 3 sets of forms are used according to the speed of sinking.

The concrete is poured in quite wet and needs very little tamping. The aggregate consists of ordinary boulders broken to 4-inch maximum size. The cement is mixed in 1-3-5 proportion. The reinforcement consists of 1-inch plain square bars. Vertical bars are placed 6 inches from the outer edge at 18-inch intervals. Every 3 feet horizontal, circular bars are wired to the vertical rods. Horizontal rods are also placed across the corners, 6 inches from the edge, at 3-foot intervals.

The concrete is mixed in a half-yard mixer and hoisted in a bucket swinging from the derrick. Three to four men on the wall will dump and tamp the concrete for a 5-foot section in a shift, attending to other necessary work on the wall. Before fresh concrete is added, the old wall is thoroughly washed. The wall is kept about 20 feet above the shaft collar to give sufficient sinking weight. To keep it higher than this would increase the weight and might induce excessive strains in the concrete at the collar of the shaft. If additional weight is needed, the concrete section may be made thicker on top, and, as it reaches the surface, the enlarged section may be trimmed to the desired dimensions.

The first 30-foot advance was made in the ordinary way by loading into a bucket suspended from the derrick. When water level was struck, a clam-shell bucket was used down to the 60-foot point where boulders became so thick that it became evident that air pressure could not be dispensed with. Preparations for this were made as follows: At 60 feet above the shoe-point an air-tight wooden platform was built, made of 4 layers of timber notched in the concrete; all cracks, bolt-holes, etc., were caulked with oakum. In the center a $3\frac{1}{2}$-foot hole was left through which a steel cylinder passes attached to the bottom of a 5-foot (diam.) Moran air-lock. At a depth of 60 feet the air pressure is 15 pounds, while at 100 feet 30 pounds will be needed.

When handling men, one man can take care of the air-lock. It takes a man from 2 to 5 minutes to go through the air chamber according to his physical condition. A bucket of muck goes through in the same way only the equalization of pressure is quicker, about 1 minute being needed. Two men are needed at the lock when hoisting. The wooden platform is covered with water to a depth of 6 inches as a fire protection. This layer of water also indicates the leakage of any air. The rate of progress for the first 60 feet was approximately 20 feet per month, three 8-hour shifts being employed.

Costs.—The cost of sinking under difficulties varies with conditions. It is currently stated that the contract price charged by the Foundation Company for sinking through overburden and sealing the shaft in the ledge is $500 per foot. The cost of sinking in the ledge varies from $100 per foot under fairly favorable conditions to $250 and $300 under heavy water flow.

Plate V illustrates shaft pockets, pump stations and pumps at a large mine.

THE SYRACUSE SHAFT

The extreme difficulties surrounding shaft sinking in some localities on the Missabe Range are well illustrated by the record of the Syracuse shaft. The Syracuse ore-body is partly under and partly adjoining a good-sized lake. The latter portion is overlaid by 90 feet of water-logged ground, the lower 40 or 45 feet of which is quicksand. Toward the end of 1904 a first attempt was made to put down a shaft by a series of telescoping coffer-dams made of wooden sheet piling in 18-foot lengths. The ground contained within the first coffer-dam was excavated very slowly—the pumps could hardly handle the influx of water and sand. When the attempt to drive the second set of piles was made, a bed of stone was encountered. The increasing inrush of water and sand together with the difficulties met in pile driving caused the company to abandon the shaft after four or five months of expensive and useless work.

It was decided to attempt another shaft at a point some 200 feet away under a heavier overburden, where the probability of encountering boulders was considered less. This entailed the expense of moving the permanent surface plant from the first shaft up a steep hill to the new location. The new shaft was started on "company account" in the ordinary Missabe style—suspended sets, 3-foot centers, hung from a large round collar set, with 2 by 10-inch vertical planks slipped behind the sets. This shaft was sunk to the quicksand, the excavation being large enough to admit a sheet pile coffer-dam with which it was expected the quicksand could be readily traversed. In November, 1905, pile driving was started, a 3,000-pound hammer and the heaviest sheet steel piling obtainable being used.

Upon completion of the coffer-dam, a sinking pump was hung in the shaft, and the excavation begun inside the coffer-dam, the sides of which were braced by timbers at 3-foot centers. After the first 20 feet progress was slow and increasing trouble was experienced from quicksand boils in the bottom followed by movement and distortion of the timbered upper section of the shaft. This evidently was caused by a break in the sheet piling and it was hoped that a drop shaft of 10 by 10 timbers could be forced down inside the piling to shut off the leak. The crib stuck after going down a few feet and could not be budged. Meanwhile the surface at the shaft collar was continually caving away. The cave was dumped full of hay and straw, in the hope that this material would find its way downward and possibly stop the leak.

The surface subsidence caused the collar-set timbers to settle and the entire wooden section of the shaft began to telescope the coffer-dam. A bearing-set of heavy rails and timbers was thrown across the top of the coffer-dam to support the timbered section of the shaft and prevent further subsidence and telescoping. After this a final attempt was made to stop the coffer-dam leaks by forcing sheets of iron alongside the sheet piling and by pumping grout down to the ledge through a 2-inch pipe which was shifted frequently. After the grout had set, a few feet of advance was made, whereupon the sand and water again broke in and boiled up. The management were thus convinced of the futility of any further attempts to put the shaft down by ordinary methods.

Early in October, 1906, the Foundation Company of New York took hold, contracting to put the shaft far enough into the ledge to make a water-tight joint, the work to be done with air pressure. It was decided to line the timbered portion of the shaft with reinforced concrete, leaving two hoisting compartments 4 feet 6 inches by 6 feet 1 inch, and a central pipe and ladderway 3 feet 6 inches by 6 feet 1 inch, and to build a concrete roof over the top of the sheet steel piling of the coffer-dam. From this point on the shaft was to be put down under air pressure, inside the coffer-dam, down to the solid ledge below, lining with 10 by 12 cribs, joints to be thoroughly caulked.

The old timbered section proved to be very unsafe due to the twisting and racking to which it had been subjected. It was found necessary to do considerable repairing and also to relieve the steel piling of the weight of the timbered shaft by suspending the latter by heavy bolts from a collar set made of extra heavy 60-foot timbers. This accomplished, the concrete lining was started from the bottom of the timbered section. Great care was needed properly to cut out and shift the struts and braces that held the shaft from squeezing in. The concrete was deposited in 2-foot layers, the timbers meanwhile evidencing the strain by much creaking and some movement. Temporary braces and jacks were freely used and progress was slow. Many braces were temporarily concreted in. After the concrete had set, these were dug and chopped out, the space being filled with concrete. Horizontal rings of 5/8-inch square iron were spaced 12 inches apart vertically in the concrete for reinforcement. Provision was made for holding the proposed concrete roof by leaving a 6 by 9-inch recess 12 inches from the bottom of the side- and end-walls.

Progress was very slow; labor was inexperienced and easily frightened away by the apparent danger; by the middle of November the lining was complete. The shaft was finally pumped out to a point about 18 feet below the location for the concrete roof and this portion was carefully relined with 10 by 12 cribs. The greatest care was required in taking out the old timbers before substituting the new. Then the form for the concrete roof was constructed. A number of 12 by 12 timbers were extended across the finished concrete walls above and the form was suspended from these by eight 1½-inch bolts. The roof consisted of a concrete slab 5 feet thick, extending into the aforementioned recess. The reinforcement consisted of two layers of 1¼-inch steel bars laid about 6-inch centers, the bars in one layer crossing those in the other. Eight 1½-inch steel hanging bolts were built into the roof to which the weight of the cribs below was transferred.

A section of 48-inch air-shaft was built into the roof and continued up to a height of 30 feet. To the top of this air-shaft a Moran air-lock was connected. By the middle of December, just two years after the first attempt was commenced, the air was turned into the shaft. The first obstacle encountered was the old timber crib. This was cut out and sent to the surface. The shaft was lined with 10 by 12 cribs, let down ready framed through the air-lock. The successive sets were suspended from the timbers immediately above them by 2¼ by ½-inch iron straps 24 inches long, fastened to each timber by two 7 by ¾-inch lag screws. Upon completion of 10 courses

of timber, iron straps of sufficient length to cover ten sets instead of two were fastened to the timbers and also to the roof bolts; in this way the shaft was secured.

For the first 25 feet it was found necessary to cement an irregular space of 3 to 6 inches left between the outside of the timbers and the inside of the steel piling. This space was filled with a rich concrete. Putting this in at the bottom instead of pouring from the top offered some difficulty. Another obstruction of frequent recurrence was the striking of loops and circles of curled up piling. The steel piles landing on stout boulders were deflected from their course, curling up under the hammer into circles often 5 to 6 feet in diameter. These were cut off with pneumatic tools. The cutting often had to be done 14 to 15 inches under water and sand to allow placing of timbers. In all it is said 145 such cuts were made.

Cutting out four piles near the top of the coffer-dam left large open spaces for the escape of air. This, coupled with the loose nature of the ground, made it very difficult to keep the water and quicksand low enough to make room for another round of timbers. The air-jet and jack-screw were in constant demand. Some time was lost through a compressor breakdown and on another occasion by a fire caused by a miner's candle.

On March 12, 1907, the ledge was reached. It was found so badly seamed and broken up that the excavation was continued 12 feet into the ledge, where a tight joint was finally made, the air being turned off May 10th. The walls of the shaft were considered tight and in good condition; the influx of water from the center of the ledge seemed very small. The contractors assumed that they were turning over a satisfactorily completed job. The following account of the completion of the shaft is compiled from data furnished by F. P. Botsford, E. M., who was in charge of the work.

The shaft was taken over by the Mining Company on June 6, 1907. The equipment at that time consisted of two 72-inch by 18-foot horizontal tubular boilers, a hoist, one small straight line air compressor and a few small pumps. One week was consumed in removing the necessary steel bolts and cleaning out the loose concrete from the bottom of the shaft. As the last of the concrete was being removed a boil of quicksand occurred, filling 4 to 5 feet of the shaft. By the use of sand-bags and plank this was passed through. The sand was found to be coming through crevices in the bottom. These were plugged with rags and wedges.

There were now no bearers in the bottom of the shaft, so the first job after plugging the sand crevices was to put in bearers and pick up the shaft. An 18-inch wide rock collar or shelf was left to support the shaft timbers; the center of the shaft was then gradually deepened 8 feet. At a point 6 feet below the bottom set a place was excavated for the bearers under the west end of the shaft. The rock was taken out from the corners between the bearer and the west end timber to make room for the studdles. These were put in, the shelf of ground remaining between them and the bottom end-timber was removed, and the wall lagged with 2-inch plank.

The placing of the east bearer was complicated by an outbreak of quicksand occurring between the bearer and the bottom end-plate. The shaft was recovered and

the hole plugged with rags and wedges. The timbers were caught up and the east end of the shaft was lined with ½-inch tank-steel plates. The shaft when completed to the bearing timbers was making about 500 gallons of water per minute which was handled by two Prescott Sinkers. This was early in August. When sinking was resumed, the first blast brought in an increase of 1,000 gallons of water per minute. This caused further delay pending the arrival and placing of a third sinker. Meanwhile another boiler was being installed. From this time until October 19th every blast brought in more water, calling for more pumps. Very little progress was made. On October 19th the shaft proper was 101 feet from the collar with a central sump 5 feet deep, making 2,700 gallons of water per minute. The following is a digest from the Superintendent's weekly reports:

October 26, 1907. Progress for the week, 2 feet. Water, 2,700 gallons. Cracking of a 6-inch discharge pipe caused some delay. Water in the old shaft lowered 6 inches this week. An old well and a spring near by have gone dry. It would seem that we are slowly but surely draining the surface, which is encouraging.

November 2. Progress 1 foot. Water, 2,700 gallons. Delays caused by poor firemen and pumpmen. Greatly hampered by scarcity of competent labor. Many makeshifts necessary to keep shaft going at all. Much sand coming in with each blast. Trying to get another set of timbers in place so we can cut off the sand with additional blocking. Water in old shaft lowered 4 inches.

November 9. No progress. Two suction hose broke this week. New set of timbers in place. Blocking effectually cuts off sand. Water in old shaft lowered 4 inches.

November 16. No progress. Thursday noon an elbow on the blow-off pipe on the middle boiler broke off. The cut-off valve was above the damaged pipe and it was impossible on account of the escaping steam to cut this boiler out of the battery. Water supply gave out and fires had to be drawn; at one o'clock the boilers were dead. Water in the shaft rose 47 feet. Pumps were recovered and water beaten down at 6 a. m., Wednesday. Balance of week consumed in repairing pumps and lengthening all suctions. The day the shaft drowned the water rose 6 inches in the old shaft. During the week it was lowered 9½ inches, a net gain of 3½ inches for the week.

November 23. Progress, 1 foot. Total depth, 105 feet. Water, 3,200 gallons. Pumps working poorly above their normal speed, many delays due to breaks, etc. Water in old shaft lowered 4 inches. Gain to date, 24½ inches.

November 30. No progress. Water, 3,200 gallons. The following notes from daily reports will explain the situation:

> November 24. 24 hours packing and overhauling pumps.
>
> November 25. 3 hours drilling; 4 hours blasting and hoisting; 17 hours packing and spragging pumps and replacing broken cross-heads on valve stem.
>
> November 26. 6 hours drilling; 5 hours blasting and hoisting; 13 hours' delay caused by pumps blowing gaskets.

November 27. 7 hours drilling; 7 hours blasting and hoisting; 3 hours' delay caused by irregular steam pressure; 7 hours changing and repairing steam line.

November 28. 13 hours drilling; 13 hours blasting and hoisting; 4 hours' delay caused by irregular steam.

November 29. 3 hours hoisting; 21 hours placing new pump valves and repairs.

November 30. 4 hours drilling and hoisting; 17 hours repairs; 4 hours' delay due to irregular steam.

The repair time taken out includes the actual repair time and the time necessary to beat down the accumulated water after starting the pumps again. This often requires several hours. Often, after lowering the water nearly to the bottom, a spring or other pump part would break and the water would rise to its former level. New boilers are being installed which should be ready to fire up in two weeks. This will remedy the irregular steam pressure. The Prescott Sinkers in use are built to operate at 65 strokes. They are averaging nearly 100 strokes per minute. This accounts for the many breakdowns. Water in the old shaft lowered 4 inches.

December 7. Last shaft measurement was 105 feet. Ten feet of water in the shaft since Thursday night when a patch on one of the boilers leaked and the fires had to be drawn, only two boilers being left in commission. These will just hold the water at the pumps. Expect to complete repairs by Wednesday. Water in the old shaft lowered 2½ inches.

December 14. Progress, 1 foot. Patch on boiler repaired in time for Thursday morning shift. To-day a new boiler was set up to run a Prescott Duplex Sinker placed in the old shaft. It is hoped that this will relieve the new shaft. Water lowered 3½ inches.

December 21. Progress, 2 feet. One set of timbers put in. Some low grade soft ore in the northwest corner at a depth of 108 feet. The pumps have reached their suction limit. Five pumps must be lowered next week. Pump was started in the old shaft Monday at noon and beat down the water 15 feet. Surface water measured in the test-pits in the old tunnel was lowered 1 foot this week. Water in the main shaft decreased 700 gallons. Now pumping 2,700 gallons.

December 28. Lowered pumps 9 feet. Still have 3 pumps to lower. Holidays interfering with progress. Test-pits show water lowered 10 inches. Water level is now 6 feet 4 inches below the lake level.

January 4, 1908. Much time spent in repairing and lowering pumps. Three pumps now in their new position. Test-pit level lowered 1 inch.

January 11. On Thursday all the pumps were in place. To get the suction on the upper pumps it was necessary to run the suction through the dividers into the adjoining compartment and carefully pass it between a lot of steam pipes. This took time. Meanwhile two of the bottom pumps gave out and the water immediately rose to the level of the upper pump. The two remaining pumps could not hold the water. Steam was shut off and the shaft filled. Friday a sixth pump was placed in the shaft

and by 10 p. m. the upper pump was recovered. To-day was spent in cleaning up 3 weeks' accumulation on the bottom of the shaft. This material was hoisted by hand in bags, no room for a bucket with the sixth pump in the shaft. As soon as the pumps are in good working condition, the new pump must be hoisted to enable us to get out the broken pump below it. When this is out, the new pump will again be lowered.

Present indications tend to contradict the assumption that the water we are pumping comes from the lake, but suggest rather that we are draining the bulk of the valley in which the Syracuse shaft is located, some 12 square miles in area.

January 18. Progress, 1 foot. Total depth, 109 feet. Water, 2,600 gallons. No. 3 test-pit water lowered 1.2 feet. No. 4 test-pit recently started 70 feet north of shaft in a good position for a timber shaft is still dry at 55 feet.

January 25. Progress, 1 foot. Water, 2,600 gallons. No. 3 test-pit water lowered 0.41 foot. No. 4 test-pit struck water at 80 feet, a distance equivalent to 70 feet in the main shaft.

February 1. No progress. Many breakdowns with the pumps. No. 3 test-pit water lowered 0.25 foot. No. 4 half a foot.

February 8. Depth still 110 feet. Starting a new 6 by 6 compartment on the west end of the shaft to give more pump room. Have drifted 3 feet in this compartment this week and have 3 feet to go. Pit No. 4, 80.66 feet to water. The ledge in this pit is 100 feet below the collar, leaving 19.34 feet of quicksand over the ledge at this point.

February 15. New compartment excavation completed. Bearer hitches cut. No. 4 pit, 81.91 feet to water, a gain of 1.25 feet.

February 23. Two bearers (13½ and 11½ feet) placed in the new compartment. Progress, 1 foot. No. 4 pit, 81.58 feet to water, a loss of 0.33 foot. Pumping 2,600 gallons.

February 29. Progress, 1 foot. One set of timbers placed. A No. 10 Cameron Sinker placed in new compartment and partly piped to surface. Pumping 2,700 gallons from main shaft. No. 4 pit, 81.75 feet to water, a gain of 0.17 foot. Still have 18.25 feet of quicksand over the ledge at this point.

March 7. Progress, 3 feet. New pump in commission. Raising 2,700 gallons of water. No change in water level.

March 14. A spliced 13-foot bearer and an 11-foot reinforcing bearer were placed in the east end of the shaft. One set of timbers in and blocked. The west compartment was cut down 6 feet.

March 21. Water-ring and 5½ by 5½ by 4-foot tank located in the new compartment. A second Cameron pump was placed at this point. Both Camerons taking their suction from this tank.

March 28. Progress, 2 feet. Piping second Cameron completed. During the month of March our gain was 5 feet—one set of timbers and two bearers placed—6 feet of ground excavated under the Cameron pumps in new compartment. A water-ring placed around the shaft and a tank built in the new compartment.

April 4. No progress. Depth, 117 feet. Very little chance to work on the bottom. Heavy pump repairs. Pumping 2,900 gallons.

April 11. Monday and Tuesday, pump repairs and lowering No. 1 Prescott. Wednesday, No. 3 Prescott was lowered 12¾ feet. Friday, No. 2 Prescott was lowered 12½ feet. Saturday, the men worked uninterruptedly on the bottom. Pumping 2,900 gallons.

April 18. Pumps working without delays. Progress, 3 feet and one set of timbers. Stopped No. 1 shaft pump on the 14th. On the 15th the water in the test-pits had risen 13 inches and our flow in the main shaft increased by 250 gallons to 3,150. Pumping at No. 1 shaft will be resumed at once.

April 25. Progress, 3 feet. Five days' pumping at No. 1 lowered test-pit water from 8 to 10 inches, proving the advisability of keeping this pump going. Water level in No. 4 pit is 4 feet from the shaft collar, leaving 16 feet of quicksand over the ledge at this point.

May 9. No progress. Lake rose 26 inches Wednesday morning; we decided to break the log-jam ourselves and tackled the job with five miners. On Saturday night over a million feet of logs had been moved through to the lower lake and a good channel was open along the entire river. The lake level dropped 16 inches by Saturday night. Pumping 3,200 gallons of water.

May 16. Progress, 2 feet. On Monday No. 1 Prescott was lowered 18¼ feet. Heavy rains and no attention paid to log-jam by the Lumber Company caused the water to rise steadily in the lake. Pumping 3,500 gallons.

May 23. Progress, 2 feet. Total depth, 130 feet. Lake high all week. Pumping 3,500 gallons of water.

May 30. Cleared logs Monday and Tuesday. Lake rising. No work possible in the shaft. Cleaned a boiler Sunday; it was impossible to keep up sufficient steam with three boilers. Water gradually rose and covered the pumps in the shaft. No. 1 boiler started to leak at the end of the flues and was shut down Monday for repairs. Monday night we recovered the pumps with the help of steam furnished by a locomotive. No shaft work. Pumping 3,700 gallons of water.

June 6. Lake slowly lowering. Pumping 3,500 gallons. In No. 4 pit attempt to locate the ledge with a churn drill. Pit is 85 feet deep. At 104 feet struck a boulder. At 106½ feet the drill was in hard ground but could not bring up a boring sample.

June 13. Lake rising, due to heavy rains. One Prescott broke down. So far unable to get another pump. Making 3,700 gallons of water.

June 20. Straining every pump to its utmost to keep down water. Two Prescotts broke down simultaneously. Others could not hold the water so the cripples could be repaired. Water rising in the lake. Decided to shut off the steam with four Prescotts in good working order. When the high water has receded, we can beat down the water with another Prescott, and recover submerged pumps. The crowded condition of the shaft will make this difficult. It will mean 7 Prescotts and 2 Camerons, nine 6-inch columns, nine 3-inch exhaust pipes, and 7 steam lines in a 6 by 13½-foot shaft.

June 27. Still raining, though the lake is going down a little daily. The boiler plant is being overhauled preparatory to unwatering as soon as conditions warrant it.

July 4. The ninth pump was placed in the shaft this week. It was difficult to find an opening large enough to place a 6-inch blower down the shaft. Got the blower in after two days' work. The blower is run by compressed air; it will throw 400 to 500 gallons of water per minute. Expect to start pumping on the 7th.

July 11. Started pumps Tuesday. By Wednesday noon the water was lowered 22 feet. No. 8 Prescott stopped Wednesday, 3 p. m. Stopped pumping. Hoisted the blower and No. 9 Prescott. Then hoisted No. 8 out of the water. A key had slipped out of a short crank. Pumps overhauled and replaced. By removing the piping from Nos. 3, 4, and 5 we could crowd in another Prescott low enough to cover No. 5. This would be a great advantage in beating down the water.

July 18. No. 10 pump was placed and started July 14th. Twice this week Nos. 8 and 9 were hoisted for repairs. Gaining slowly but surely. The old dump was cleared away in preparation for starting a timber shaft 500 feet from the main shaft. The bearers are in and the sinking shoe is being put in place. (This shaft is separately described on page 109.)

July 25. Steady rains all the week. Pumping 3,800 to 4,000 gallons. To-day succeeded in pulling up No. 2 Prescott, one of the lowest pumps. The turnbuckle on the bridle broke just as we got the pump to the surface of the water. This is repaired and the pump is piped to the surface. We now have 5 Prescott pumps and the blower above the water giving a 4,000-gallon capacity. The heat in the shaft is intense. The steam generated by water falling on the hot pipes and cylinders makes it impossible for a man to stay in the shaft an hour at a time. This seriously hampers the work of lowering the pumps.

August 1. Pumping steadily all week at full capacity, lowering pumps as they require it. Uncovered the Camerons at the 107-foot level on Friday. It will require several days pumping at this level to make an impression on the water.

August 8. Four of the Prescotts were lowered 10 feet each this week. No. 7 Cameron is repaired and running. No. 6 Cameron is being repaired. To-day the water level is down to 110 feet. Pumping from 3,700 to 4,100 gallons. Pumps are very close together so that all repair work is slow and difficult. The test-pits indicate gradual subsidence of surface water.

August 15. Sunday night Nos. 2 and 3 Prescott broke down and were pulled to the top of the water for repairs and lowered again.

August 22. Repairing pumps all week. Cramped place and intense heat make this work very slow.

August 29. Tuesday night No. 6 Cameron and No. 2 water column broke. Raised No. 2 Prescott for repairs. No. 1 exhaust needed repairs. No. 4 Prescott also. At present we are running 1, 2, 3, 4, and 8 Prescotts; 6 and 7 Camerons. No. 5 Prescott is being repaired.

September 5. Just succeeded in keeping the water below the bottom pumps this week. Much repair work.

September 12. Repairing pumps all week. Some of the bottom pumps are in such poor shape that it is necessary to replace every movable part with a new one.

September 19. Pumping 3,100 gallons and holding 3 feet above the bottom of the shaft.

September 26. Hoisted 20 bags of dirt from shaft bottom to clean the wind bores of the pumps.

October 3. Hoisted out Nos. 8 and 9 Prescott pumps; this gave room to use a bucket in the shaft, so that we were able to clean up the shaft bottom. Pumping 3,100 gallons.

October 10. Air chambers placed on the seven water columns. General pump repairs.

October 17. Shaft bottom advanced 2 feet. Total depth, 131 feet. Pumping 3,100 gallons.

October 24. Progress, 1 foot. One set of timbers placed. A drift started on the east end at 132 feet was driven 5 feet. In this drift two Prescott sinkers will be installed.

October 31. Drift advanced 4 feet and two 14-inch bearers placed in the drift to support the pumps and drift sets. Bad coal and the cleaning of boilers caused delays. While one boiler was being cleaned the remaining four could not maintain sufficient steam to keep the water out of the shaft.

November 7. Progress in the shaft 2 feet to 137 feet. Drift advanced 3 feet to 15 feet.

November 14. Progress, 1 foot. Drift advanced 3 feet. Nos. 8 and 9 Prescott were lowered down the shaft and placed in the drift. Pumping 3,100 gallons.

November 21. Progress in the shaft, 2 feet. No. 9 Prescott is piped and running. No. 8 discharge column being placed. Pumping 3,100 gallons.

November 28. Progress, 1 foot to 140 feet. No. 8 all piped and ready to steam up. Cleaning boilers all the week. Many delays in the shaft on account of low steam.

December 5. Progress 2 feet and one set of timbers.

December 12. Depth of shaft, 142 feet. Much trouble keeping water out of the bottom because the pumps were hung too high. Thursday, No. 4 pump was lowered 20 feet; while we were piping up No. 4, No. 1 pump lost a valve on one side, halving its capacity. Simultaneously No. 8 lost a valve-head-pin and No. 6 blew out a spindle cap. One pumpman went to the surface for a pin for No. 8; he was followed by another pumpman in search of a spindle cap. The water raised rapidly and No. 8 was submerged before they could return with the pin. The spindle cap was placed on No. 6 in time to start it up and check the rising water. No. 4 was pulled back with chain blocks to its original position in order to have its pipe lines connected up. While this work was in progress No. 6 again blew its spindle cap, No. 7 blew off its air chamber, and No. 9 ceased throwing water. No. 1 was doing very little, while Nos. 2 and 3 were working at full pressure. The series of accidents enumerated above all happened within a single hour. Thursday night an

old Prescott (No. 10) was lowered. On Saturday another new Prescott (No. 11) was lowered. After a few hours No. 10 suddenly stopped and both Nos. 10 and 11 had to be pulled to the top of the water to repair 10. On being replaced, No. 10 worked for three hours and stopped. No. 10 is an old pump so we are overhauling it carefully and lowering No. 11 ahead of No. 10. Nos. 2 and 3 are holding the water at the 75-foot level. If they hold out until Nos. 10 and 11 are in place, the four pumps together will uncover the submerged pumps. Working day and night to accomplish this.

December 19. Recovered the top pumps Wednesday night and bottom pumps Thursday night. Spent Friday and Saturday packing and placing new valves.

December 26. Steam and water lines needed repairing. Simply holding water.

January 2, 1909. Holding water and completing repairs.

January 9. Hampered by extreme cold—25° to 45° F. below zero. Started new 5-inch steam line from boiler plant. Started work in shaft bottom.

January 16. Progress, 2 feet. Completed 5-inch steam line down the shaft and connected up all but 3 pumps. Pumping 3,100 gallons.

From January 16 to February 20 the average weekly progress was 3 feet. No special mishaps. On February 20th the shaft was down 160 feet to the main level.

From February 20th to March 20th the shaft was deepened to 167 feet, heavy bearers were put in, and the shaft station on the 160-foot level was cut a total distance of 28 feet completed and timbered.

From March 20th to April 24th three of the pumps were gradually moved down to the station, connected, and piped up. Ladders were looked after, wire guy ropes put in, and the small buckets were replaced by two 1-ton buckets. Two drifts were started. The south drift reached 25 feet and the east drift 8 feet.

From April 24, 1909, the work through the main shaft was confined to drifting and cross-cutting, cutting pump stations, etc., incident to opening the 160-foot level.

On September 4, 1909, there had been completed the following work:

Drifting and cross-cutting, 200 feet.

A pump station 18 feet by 83½ feet long.

A condenser chamber 12 feet by 18 feet long.

A sump for the pump station 16 feet deep, terminating in a 24-foot sump drift.

"A" and "B" station pumps installed and working.

No. 1 drift had cut the ore-body at 43 feet. From the timber shaft 4 drifts were run out on the "A" sub—No. 1 north 94 feet, No. 2 east 72 feet, No. 3 west 20 feet, No. 4 north 23 feet; total, 209 feet.

The ore shipments to date aggregate 67 cars varying from 58 per cent to 62.5 per cent iron and from 0.020 to 0.050 per cent in phosphorus.

From the commencement of No. 1 drift on the sub-level a heavy stream of water flowed from the back through the lagging. One day a little sand was noticed coming through the water. Immediately preparations were made to dam up the entrance to the drift in order to confine any possible sand to it. Shortly after this, No. 1 drift

filled with sand and the cave went through to the surface. This was the first indication of what might be expected in the working of the mine.

During September and October various drifts were advanced, about four crews being engaged in drifting. Frequently drifts were temporarily stopped on account of sand rushes. During this time 4 Prescott sinking pumps were installed to act as relays in case of emergency. Four water columns and two No. 10 Cameron pumps were taken out of the shaft.

On November 5th No. 6 drift on *A* sub blasted into quicksand and the men were beaten back through three successive gates before they succeeded in stopping the sand rush. No. 6 drift was filled with sand its entire length (70 feet). No. 2 drift (190 feet long) was filled for 90 feet from the breast. Sand poured down the raise from the main level filling the sump and skip-pit and covering the main level 32 feet below with a 3-foot layer of sand. The pumps were handling 4,000 gallons and it was only by the greatest exertion that they were saved. It took two weeks' work during which time some 800 buckets of sand were hoisted to clean the mine.

The superintendent's report for December 11, 1909, states: "The shaft house was finished last week, skip-guides were placed in the shaft, dump trestle built, and mine thoroughly cleaned out. Regular drifting was resumed this week, No. 2 drift being advanced 10 feet to 124 feet, the temporary track on main level was replaced by 20-pound rails. All sinking pumps were removed from the timber shaft. Pumping 4,000 gallons. All necessary preparations for handling ore are completed."

Drifting was resumed and continued in six faces without interruption until December 22d, at which time the three bottom holes in No. 2 drift (151 feet in) showed sand mixed with the ore. No pressure and no additional water. Two heavy gates were installed and the drift stopped pending investigation of conditions ahead.

From then until the end of February slow progress was made in some four or five drifts on the main level, the pumps handling from 3,500 to 4,000 gallons. Constant observations were made on the water level in the test-pits before referred to and the reports indicate a very slow gradual lowering. Occasional sand runs are noted.

The report for March 5th contains the following:

"On March 1st at 4 a. m. sand and pebbles broke through the back of No. 5 drift close to the breast. The hole was blocked and the back lagged up but the water continued to come through with great force, forcing the sand through the openings between the lagging. The sets near the breast were showing evidence of considerable weight. A strong barricade of heavy timbers and plank was then built 35 feet from the breast. Two additional barricades were started and partly completed when, with a roar, the quicksand broke into the drift and went through the barricade. The water and sand continued to rush through the drift toward the shaft carrying everything before it. There was nothing to do but shut the two emergency gates in the main drifts, close to the shaft. These were closed with all possible speed, but not before considerable sand had come through with the water. All day sand came through every crack and crevice, filling the sump and skip-pit. There was a foot of sand on the pump station floor and 3 feet of sand in the drifts and shaft station. At 11

p. m. the water began to run clear and by morning it was running practically free from sand. We must have pumped several thousand tons of sand during the 20 hours. The drifts behind the gates were evidently full of sand. In No. 1 raise, which is 140 feet from the shaft, the sand was up to within 15 inches of the caps and the water occupies the balance of the space. It has been a struggle to keep the water out of the mine, and several times we have come very near losing the pumps. To-day we are lowering a 14 by 8 by 12 sinker in the timber shaft to aid the main shaft pumps. We are pumping 4,700 gallons of water per minute, which means we are lifting 28,000 tons 160 feet every 24 hours."

The following month was spent in pumping and cleaning out sand from stations and sumps and measuring the water level in the test-pits. The report for the week ending April 9, 1910, states: "Since March 1st the water flowed through the west gate. On Thursday it suddenly changed its course and flowed through the east gate. Much fine sand was stirred up and the drifts we cleaned out a few weeks ago are again filled, the sand being up to the wind bores of the pumps. The rainfall has been very light this spring and the snow melted so slowly that the river is lower this spring than at any time in the history of the Syracuse Mine. In spite of this favorable condition and the 4,000 gallons we are pumping, the water is slowly rising in the test-pits." Subsequent reports are of the same tenor. The last week in May it was decided to pull the pumps and abandon the mine.

Prior to abandoning the property the question of salvage of the two station pumps was carefully considered. It was estimated that one station pump and condenser and all steam and water piping could be removed at a maximum cost of approximately $5,000, requiring 22 days' time. This would leave the other station pump in the mine stripped of its oil pump, gauges, steam traps, etc. This plan called for the lowering and piping up of 4 sinking pumps. It was estimated that this would take 10 days, 180 shifts of shaftmen, and 80 shifts of pipemen.

The alternative plan was to use both shafts with a maximum of 10 sinking pumps and remove both station pumps. The time estimate on the plan was 72 days, exclusive of 30 days waiting for pumps and supplies to be ordered immediately upon receipt of instructions to close down. The estimates were detailed as follows:

LABOR—		
Labor during 30 days' wait	$1,831.80	
Direct labor, 2,242 man-shifts	6,316.05	
Indirect fixed labor	4,386.52	
Total labor		$12,534.37
SUPPLIES—		
Coal	$11,271.00	
Mine supplies	1,400.00	
Pump supplies	1,300.00	
Total supplies		13,971.00
TOTAL LABOR AND SUPPLIES		$26,505.37

The estimated maximum cost of this plan was $26.505.37, to which must be added the depreciation on 5 new Prescott sinkers at $5,400 and $1,000 worth of piping needed to carry out this plan.

The first plan was adopted and successfully carried out at a cost of $2,553.34, much below the estimated maximum. Of this, $1,618.16 was for labor and $751.32 for coal. The time occupied was from night shift of June 1st to June 18th, three 8-hour shifts working without intermission. The crew varied with the nature of the work, varying from 40 to 95 men per 24 hours. All shaft work was run on 3 shifts, 8 hours each.

A typical division of the labor during the heavy sinking work is as follows:

8- hour men—
- 3 to 4 miners and 1 pumpman on each of 3 shifts.
- 1 fireman on each of 3 shifts.

10-hour men—
- 3 pipemen on each of 2 shifts.
- 5 laborers (surface) on each of 2 shifts.
- 1 blacksmith and 1 helper on 1 shift.
- 1 machinist on 1 shift.

12-hour men—
- 1 hoisting engineer and
- 1 lander on each of 2 shifts.
- 1 captain on 1 shift.
- 1 foreman on 1 shift.
- 1 head-pumpman.
- 1 boiler cleaner.

The company's records show the following average number of men employed per 24-hour day, for the two periods indicated:

Dates	Average number of men employed per 24 hours			Average man-shifts per 26-day month
	Surface	Under-ground	Total	
Shaft sinking period July 1, 1908 to January 31, 1909.........	42.3	22.7	65	1682
Development period February 1, 1909 to December 31, 1909...	41.0	47.0	88	2384

The total coal consumption charged against the Syracuse shaft during the two heaviest years (from January 1, 1908, to January 1, 1910) is 22,563 tons. This means an average of practically 940 tons and a cost of from $3,500 to $4,000 a month.

It may appear to the casual reader that the management should have anticipated all these troubles, that time and money would have been saved had they been prepared in advance therefor.

It must be remembered that when the company took over the shaft and commenced their sinking campaign they did not anticipate tackling a wet job with two old

boilers; they had every reason to expect a water-tight shaft. They enlarged their boiler plant to 4, and subsequently to 7 boilers, as rapidly as possible.

SYRACUSE TIMBER SHAFT

On July 1, 1908, preliminary work was begun on the timber shaft mentioned on page 103. This shaft was started 9 by 8 feet. It was intended to force down a shoe in accordance with the method used at the Buffalo and Susquehanna (page 88). After a depth of 30 feet had been reached with more or less difficulty (the shoe was jacked out of place several times), this method was abandoned. On August 22d the depth was 55 feet and the sand was pulling the timbers out of line. Four 1-inch wire ropes were hung from surface trusses to help support the timbers.

On September 5th the shaft was down 82 feet to water. By sounding, hard ground was struck 12 feet below this point. As a precaution, 12-foot lath were driven as far as possible before starting the pumps. On September 19th the lath were down from 7½ to 8½ feet; a large boulder held up a few of the lath.

During the week ending September 3d a No. 9 Cameron pump, previously placed and piped, was started, the shaft was advanced 18 inches, a set of timber placed, and a sump excavator for pump suction, using a drain box as described on page 88. By running the pump slowly and taking care not to excavate sand too fast the water was kept about 3 feet below the bottom shaft set. The following is a digest from the weekly reports beginning with week ending October 10th:

On three sides the sand drains nicely to about 3 feet beneath the bottom timbers. On the fourth side it retains the water right up to the timbers and attempts to drain it have failed. Drove lath all week trying to drain the sand. Do not know just whence the water comes, possibly from the ore through a drill-hole close to the northwest corner of the shaft.

October 11. Water still hanging up in the sand on the north and west sides of the shaft: it is impossible to remove the sand under these conditions. We are cutting 3 inch plank wedge-shaped and jacking them down under the bottom timbers. We make a 3-inch gain on each four-piece set jacked in place. This week we put in five 3-inch sets making a total depth of 84 feet 9 inches.

October 24. The collar set and bearers had been in bad condition for some time and they were replaced this week. Two 3-inch sets were jacked in place. Water is still hanging up several feet above the sump in the shaft and we are moving much more sand than is safe in getting our timbers in. For this reason we stopped jacking and are driving lath again. The shaft is making so little water that a No. 9 Cameron did not work well and it was replaced by a No. 5. The depth of the shaft is 87 feet, a gain of 27 inches. Part of this is due to the shaft settling down in the sand.

October 31. Two 3-inch sets were jacked into place. Progress, 6 inches. Total depth, 87½ feet. Water is still holding up in the sand and giving much trouble.

November, 7. Driving lath and pumping all week. The lath are now down to the ledge. Shaft timbered to 87½ feet. Sounding showed the ledge to be 7 feet below the last set. Water is giving much trouble holding up in the sand back of the timbers.

November 14. Two 3-inch sets placed. Depth, 88 feet. Strain on holding irons taken off three times and collar of shaft repaired.

November 21. One 3-inch set placed. Four broken bearers taken from shaft collar and new ones put in. Some reinforcing of strained timbers.

November 28. Five 3-inch sets placed. Progress, 29 inches. Depth, 91 feet. Ledge still 3 feet below us.

December 5. Two 3-inch sets placed. Water still hanging up in the sand, and the side pressure on the timbers is increasing greatly. We are bracing the 3-inch sets with heavy false sets to be removed when the water has been drained from the sand. Struck some boulders this week that added to our troubles.

December 12. Two 3-inch sets placed. Progress, 1 foot. Depth, 93 feet. As soon as the water in the main shaft rose high enough (Thursday, 6:30 p. m.), it began to rise in the timber shaft. We strengthened the timbers in every conceivable way and abandoned the shaft. Can not see how it can be harmed. We cut the shaft loose and it is gradually dropping down. Expect to find it almost or right on the ledge when we pump it out.

December 19. Water drained down to original water level, 80 feet below the collar. About 6 feet of sand in the bottom. The shaft has settled more on the north side than on the south, but is generally in good shape.

December 26. Guides and dividers straightened out and collar timbers built up, to make up for settling.

January 2. No. 9 Cameron pumped out the shaft. Six feet of sand cleaned out. Shaft resting on the ledge on one side. All timbers in good shape except two 3-inch planks that had to be jacked back in place. Shaft connected to the ledge all around by wedges and false sets of timbers placed inside to hold the outer sets and wedges in place. Expect to sink in rock far enough to get in two bearers 6 feet below the present timbers. After the bearers are in the intervening ground will be removed and the shaft made to rest on the bearers. A tight joint will then be made at the contact of the sand and ledge.

January 9. Progress, 5 feet in rock. Struck a small stream of water on the south side of the shaft at 99 feet. Water still coming from the shaft above. The upper part of the ledge is broken taconite, and it will be necessary to reinforce the timbers. Hampered by a cold spell, 25° to 45° F. below zero.

January 16. Progress, 2 feet. Pipe line was run from the compressor receiver down the shaft to furnish air for rock drills.

January 23. Progress, 1 foot. East end-hitches cut and a spliced 14-foot bearer placed. South hitch cut on the west side. While cutting north hitch, broke into a churn drill hole several feet in diameter filled with quicksand and water under heavy pressure. Decided to sink deeper and carry a false wall-plate on the north side, until the drill hole drains or becomes small enough to work up to it safely.

January 30. Shaft opened to full length and deepened 2 feet. One set of timbers placed above bearers. Water increased to full capacity of the No. 9 Cameron.

January 30 to March 13. Progress during the 6 weeks, 4 feet. Water increased and now have two Camerons and one Prescott in the shaft. Two bearers placed; 5 false sets removed, shaft enlarged, and permanent sets placed. Water still hangs up in the sand and gives much trouble.

While we were attempting to connect the re-timbered lower section of the shaft with the upper section at the contact between the sand and the taconite, the sand rushed in and filled the shaft to the break, a depth of 12 feet. This happened early in the week. So far we have not succeeded in blocking the hole.

March 27. Timbers showing evidence of heavy strain. Reinforced them by placing over a hundred studdles between the sets. Trying to block sand hole by forcing concrete into it.

April 3. Sand hole finally plugged and the shaft almost cleaned out.

April 10. Shaft cleaned out and a platform built 8 feet above the bottom to prevent suction in event of another sand run.

April 17. Attempted to connect lower and upper timber sections of shaft. Got one studdle in place, but while we were cutting out for the next, the sand and water broke in and flooded us. We opened up the shaft at the contact and forced in 60 sacks of concrete. This should set and hold the ground.

April 24. The east end and half of the south side connected this week. The north side is continuous. Have been forcing concrete into the west end and also adjacent to timbers already in place.

May 1. Connection completed and shaft bottom recovered to 108 feet.

May 1 to May 29. Progress, 3 feet; for the month, 14 feet. Steam line, badly damaged by movement, renewed. Two Prescotts and one Cameron working.

June 5. Progress, 8 feet. Total depth, 130 feet. Ore encountered in the northwest corner at 123 feet, and pitching southeast. At 130 feet the shaft is half in ore.

June 12. Progress, 3 feet. Cutting hitches for bearers at 138 feet. Expecting to open "A" sub at 137 feet.

Elevation of main shaft collar..........926.1 feet
Elevation of "A" sub..................790.0 feet
Elevation of main level................758.0 feet

Main level is 160 feet below the shaft collar and "A" sub is 32 feet above the main level.

June 26. Progress, 6 feet. Total depth, 144 feet. A platform was built at 137 feet, from which a drift will start west.

From June 26 to December 11 the pumps were kept going and six drifts, aggregating several hundred feet in length were run on the 137 feet sub. On December 11th the pumps were pulled. The property was subsequently abandoned.

MINING VERY WET GROUND

One of the difficulties the underground mines occasionally have to contend with is a wet, heavy overburden consisting of boulders and much (sometimes 60 per cent) quicksand. Between the ore and overburden there is often a capping of impervious

clay that prevents a thorough draining of the ground above. Consequently when th roof is dropped, sudden, heavy rushes of quicksand are apt to come. Yet without th clap capping it would be impossible to hold the roof.

One company in the Hibbing district is operating an irregular, low-grade ore body, badly mixed with taconite, lying under 130 to 160 feet of such overburden About 6 million tons are to be mined by underground methods, while more than twic this amount is to be mined open-pit, requiring the removal of about 8 million yard of overburden at a cost of nearly two and a half million dollars.

The mine is opened in the usual way. The slices are carried one set wide wit 10-foot posts in the upper 60-foot zone. This zone is very irregular. The layer o paint-rock and clay overlying the ore is very heavy and will often crush the timber in a week or ten days.

When a room is mined and ready to cave, it is walled in tight before the timber are blasted. After the roof caves the water breaks through the cracked clay abov and is caught by the board wall, being held there by the broken and practicall "puddled" clay in the room. After several days the adjoining slice may be worked Sometimes the back fails to hold as a room is being mined, when the quicksand break through with a rush, sweeping everything before it and filling hundreds of feet o drift in a few minutes. This has happened several times in the course of the last fiv years.

There are many old drifts in the mine having 10-foot pillars around and ove them. Many of these are in process of extraction, the slices being removed in th order shown in Fig. 53, drawing the pillar back toward the shaft.

The present workings are all in the upper 60-foot zone. It is expected tha the mine will be more easily worked after the entire roof area has been caved dow to this level, below which the ore also is more uniform.

It has been planned to open the next level 100 feet below the present and to la out the subs with 12-foot posts. The subs are to be laid off in 100-foot blocks wit an ore-chute or mill in the center of each block. The pillars will be sliced towar their own central chutes, thereby lessening the tramming.

The present annual production under existing adverse circumstances is abou 300,000 tons with a crew of 218 men, divided as follows:

Surface, 38; underground, 180, of whom 124 are contractors. The tonnage pe man is about 4¾, while the tonnage per man engaged in slicing is about 8. Th average contract wage is approximately $2.75.

The cost of ore on the cars is about $1 per ton exclusive of royalty and genera office charges.

In the Embarras Lake district, at the eastern end of the Missabe, this quicksan overburden becomes a very serious question. The Syracuse Mining Company, afte spending close to half a million, abandoned its property and the same operators ar encountering serious, though not insuperable, difficulties at the Bangor Mine. It wa thought that the operations at this mine are of sufficient interest to warrant a detaile description.

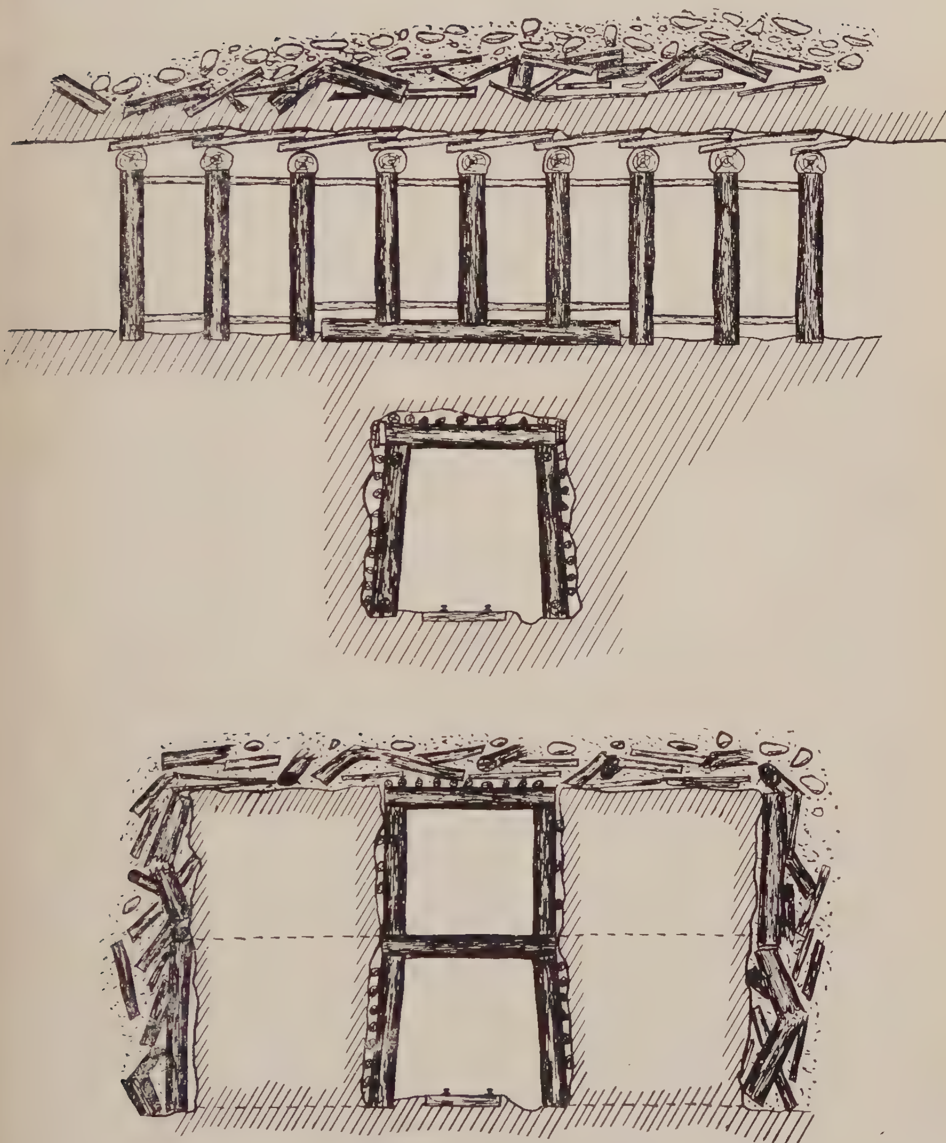


FIG. 53. Mining Old Pillars.

THE BANGOR MINE

This property is situated on the shore of Embarras Lake. The ore-body as developed is irregular in outline and covers an area of roughly 400,000 square feet. The thickness of the ore ranges from a maximum of about 200 feet to 30 feet on the edges, with a probable average thickness of 100 feet. It is estimated to contain between 2½ and 3 million tons of high-grade, non-Bessemer ore and two million tons more or less of lean ore, badly broken and mixed with rock, displaced and mixed to an unusual degree for a large Missabe deposit. The ore is overlaid by a heavy taconite capping and some 125 to 200 feet of fine sand, the average depth of the overburden is 265 feet with a maximum depth of 373 feet.

The Bangor shaft is 1,200 feet from the lake and the collar is 50 feet above the lake level. The shaft is sunk outside of the ore-body and its present depth is 350 feet. About 175 feet is sunk through sand and the balance through taconite. A large portion of the ore lies below the present main-level and the shaft will in time be sunk an additional hundred feet or more.

Water was found at 50 feet from the surface, corresponding to the lake level, and a contract was let to the Foundation Company, of New York, to sink a circular concrete shaft to the ledge, far enough to make a good joint. Their first step was to excavate a cribbed pit, 25 feet in diameter and 25 feet deep. A derrick was erected at the side of the pit, the shaft and machinery were enclosed to protect the men. A large cast-iron shoe, in sections of convenient size, was assembled in the pit; sheet iron forms were built up and fastened to the top of the shoe by bolts. These forms were built in sections that could easily be handled with the derrick; the assembled forms were circular, 4 feet high, flanged on the outside. The inner diameter was 14½ feet and the outer diameter 24½ feet, giving a 5-foot annular space for concrete walling. Four sets of forms were used at a time, and 16 feet of concrete was placed before the lowest forms had to be removed. The concrete drop-shaft was sunk 50 feet to water by men shoveling into buckets. Then hand work was replaced by a clam-shell dredge bucket which filled under water from the center of the shaft, the sides falling into the center excavation, thereby ensuring the vertical drop of the shaft. At 100 feet the sand, because of its fineness and the pressure created by 50 feet of water overhead, packed so solidly that the dredge bucket could no longer load itself.

In anticipation of this difficulty the shoe had been fitted with angle irons to hold a cylindrical steel shaft made up of 10-foot flanged sections, 10 feet in diameter. These sections were added to every time the shoe gained 10 feet; rubber gaskets were used to make a tight joint. When the dredge could make no further headway, the inner steel tube was 100 feet long and 10 feet in diameter. An air-lock was then flanged to the top of the steel shaft, and the work was continued under air-pressure, starting at 20 pounds and increasing finally to 34-pound pressure.

In order to sink the caisson the shaft bottom was excavated under air several feet below the shoe. The men then came out of the shaft, the compressors were stopped, and a large release valve was opened, which quickly released the air pressure. The

shaft would then suddenly drop a foot or two at a time. Care was exercised in doing this not to reduce the air-pressure too quickly or entirely lest serious sand-boils might result. The last 15 feet the shaft had to be loaded with 300 tons of pig-iron, after the pressure was reduced to the danger point. The shaft was then carried into the rock 9 feet, a total distance of 134 feet, where a water-tight joint was made. Thirty-four feet was sunk under air. After the air was taken off, the shaft was tight at the joint and about 300 gallons per minute were seeping through the taconite in the bottom.

The shaft was turned over to the mining company in October, 1907. From October 19th to December 21st was consumed in getting ready to sink the shaft below the concrete, placing steel, lowering pumps, piping, etc. The concreted shaft was 19 feet in diameter on top, tapering to 14½ feet at the bottom. This was divided into 4 compartments with steel sets made of 6-inch I-beams and 3-inch I-beams for dividers. Angle-iron brackets bolted to the concrete were used to hold the I-beams which project into ample hitches cut in the concrete.

The steel dividers are fastened to the main I-beams with angle irons riveted to each piece. To place one set, 7 hitches were cut in the concrete and the steel concreted in place, 14 bolt holes were drilled and the bolts grouted in place, and 15 angle irons were riveted. After the men became accustomed to the work, 5 sets were placed regularly in a week.

The shaft below the concrete was carried square and lined with 12-inch timber sets. A foot of ground was left inside the wall for its protection; then sinking was commenced; the opening was gradually enlarged to full size. At six feet below the concrete, two 18-foot bearers were placed and the concrete was carefully moiled out; regulation sets were then put in until a tight joint was made between the timber and the concrete.

At first the shaft was making 300 gallons of water per minute, handled by a Prescott sinker. Before 2 feet had been sunk a couple of paint-rock seams were struck in the taconite, so that the water was raised suddenly to 800 gallons. Launders were built around the shaft and the water led to a couple of tanks of about 500 gallons capacity. Two Prescott sinkers were stationed here. The next 70 feet was sunk with one No. 9 Cameron sinker holding the water. At 200 feet two bearers were put in. The water below this point increased rapidly. At 210 feet the drill-holes were plugged to hold the water until two Prescotts could be placed and piped. At 230 feet there were three Prescotts in the bottom handling 2,000 gallons of water and the flow into the upper tank was only about 300 gallons. There were three No. 9 and one No. 10 Cameron pumps at the upper tank.

On May 23d the Master Mechanic and the Head Pumpman were scalded by a gasket in the steam line blowing out. It was necessary to shut off the steam to get the men out of the shaft. The water rose and submerged the pumps. One sinker was disconnected, one broke down, and the third was working. By the time another pump was lowered and working, the water had risen 90 feet in the shaft. The pumps were run steadily from Saturday noon to 1 a. m. Tuesday to uncover No. 1 Prescott, when the bridle on No. 4 pump broke, dropping the pump several feet and jamming it so

solidly between the timbers that it could not be budged with a tackle. The piping snapped off. The water rose rapidly and submerged the Camerons at the 140-foot tanks. These pumps were in good condition; as the water was boiling hot it was thought best to shut off steam and not risk burning out the packing and await the completion of No. 4 boiler, then building. Meanwhile the three boilers were thoroughly overhauled.

On June 13th an attempt was made to unwater with a new Prescott, expecting that the submerged Camerons would help. When the water became very hot, the latter gradually slowed down and stopped, one at a time. On June 16th No. 6 Prescott was placed and after half an hour's pumping the Camerons were unwatered, though the head-frame broke down under the strain of lowering two pumps and piping at once. When this happened, the strain was released by putting clamps on the hoisting ropes at the shaft collar. The sheave wheels were then removed from the top of the head-frame to the shaft collar and the pumps were lowered as the water was beaten down. (Week ending June 20, 1908.) On July 1st, the shaft was recovered and on July 4th the necessary repair work and overhauling was completed. In all, 5 weeks were lost in fighting water and pump troubles.

The shaft was completed without any further troubles. At 254 feet a third pair of 18-foot bearers was put in. At 284 feet a 17-foot drift was opened. A launder and concrete dam converted this into a water reservoir.

At 314 feet the shaft was stopped and at 308 feet the station was opened up. This was done by first opening a 7-foot drift with 10-foot posts and 7-foot caps, then returning to the shaft and opening the station full width, replacing the 7-foot caps with 12½-foot timbers. On November 28th the station was completed a distance of 28 feet with 12 sets of timber in place. The average rate of advance from the bottom of the concrete to the shaft bottom was 18 feet a month. This includes all the time lost while waiting for completion of the fourth boiler, recovering the shaft, etc. In judging the speed it must be remembered that breaking ground and hoisting dirt was a very small part of the problem. It was a question of fighting a steady water flow of 2,000 gallons per minute, lowering heavy pumps, repairing and working around hot pumps and sizzling steam pipes in a contracted, steam-filled area, often so hot that men could stand it for only a few minutes at a time. Under such conditions 18 feet a month would be considered good progress.

While the shaft was sinking, the crew ranged from 40 to 55 men, divided over three 8-hour shifts. (See table page 124.)

Drifting was started the last week in November, 1908. Shortly thereafter a pump station was begun north of the shaft, by first cutting a drift which was subsequently sliced out to its full width of 17 feet. The station is 17 feet wide by 14 feet high. This station was completed March 6, 1909. It is at present supported by heavy timbers which are taking considerable weight. It is ultimately to be concreted.

The two main drifts were carried forward at ordinary speed and were temporarily stopped on March 20th pending the placing of the large station pumps in the recently completed station. The pumping plant consists of two horizontal, duplex, triple

expansion, condensing Prescott Station pumps, Missabe pattern water-end, 16 by 25 by 42-inch steam cylinders, 16½-inch plungers, and 24-inch stroke, with 12 by 20 by 18-inch independent air-pumps and condensers. Each pump has a normal rating of 3,000 gallons per minute at 33 strokes. The time consumed in the installation of these pumps was from March 20th to May 15th, at which time both pumps were running satisfactorily.

From May 15th to June 12th the underground work was still interfered with by the erection of the steel shaft-house and other construction, by the overhauling of the compressor and hoist, steam and water piping.

On June 14th drifting was resumed and continued for several weeks in mixed taconite, paint-rock, and yellow ochre. The drifts are first run quite small in anticipation of heavy water flows and caving ground; they are subsequently stripped or widened to full size, 10-foot caps and 8-foot posts, giving 8 feet 8 inches and 7 feet 10 inches between the joggles.

On September 11th the development had reached the following stage:

No. 1 drift face: 198 feet from the shaft, full width and last few feet in mixed paint-rock and ochre.

No. 1 raise: Just started in ore at the face of No. 1 drift. A sample 10 feet above the drift shows 57 per cent iron, 0.088 phosphorus, 1.46 manganese.

No. 2 drift face: 235 feet from shaft, full width. Low-grade ore in the breast.

No. 3 drift face: 83 feet from shaft. Taconite and ore indications.

A sub-drift previously started from the shaft, 30 feet above the main level is in 83 feet, in taconite.

The ore pocket almost finished. (Completed October 2.)

The work of drifting and raising was gradually extended on both main and sub-level until the water began to increase. When the water exceeded 3,000 gallons, the safe pumping capacity keeping one pump in reserve, the number of working faces was reduced and on December 4th work was begun on another pump station. This station, 47 feet long, 17 feet wide, and 10 feet high with a 25-foot drift connecting it to the shaft, was completed December 25th.

On January 1st the work of drifting and raising was again opened out. The last week in January, 8 drifts and 3 raises were advancing on the sub-level, when the water again increased dangerously and 5 drifts were stopped.

Early in February one of the station pumps suddenly stopped. One low pressure cylinder was cracked. It was decided to take no chances and to stop all advance until the third pump, a compound, duplex, condensing Prescott Station Pump, Missabe water-end, 20 by 37 by 16, ½ by 24-inch stroke, should be in commission. Work on the new pump was delayed by the non-arrival of the necessary parts and materials. It was completed and running the second week in March. The mine was now equipped with three pumps, each of 3,000 gallons normal capacity, able to handle 6,000 gallons with one pump in reserve.

Drifting was resumed, the pump handling 3,300 gallons. On April 23d there was reported a total of 1,006 feet of drifts on the main level and 1,324 feet on the

sub. Most of the sub-drifts, instead of making low-grade ore as anticipated, were in taconite, and many of them were therefore stopped. Instead of 10 to 12 gangs drifting in ore on the sub, there were only four. The main-level drifts at this time were in good grade of ore. The water, however, was on the increase, 3,600 gallons.

On June 4th the water had increased to 4,000 gallons. Drifting was progressing at the rate of only 100 feet a week.

On September 3d the total advance recorded on both levels was 4,221 feet. The main level proved so wet that a new mule barn was built on the sub and a 41-foot incline cut for mule travel from level to sub. A timber-shaft to be known as No. 2 shaft was begun. The water had increased to 4,200 gallons and development work was pushed very cautiously. Sand gates (see Fig. 49, page 119) were built in the sub-drifts to guard against possible sand rushes. The weekly progress varied from 150 to 250 feet drifting and 0 to 25 feet raising.

On January 1, 1911, the aggregate length of drifts and cross-cuts was 6,685 feet. On May 20, 1911, this had been increased to 11,000 feet. About the middle of May a test-raise was put up in drift No. 11 on No. 5 sub, to ascertain how the sand over the ore-body would act in the regular slicing and caving work to follow. The test-pit data indicates 150 feet of wet sand over the ore at this point. The raise showed: 8 feet merchantable ore; 5 feet paint-rock; 4 feet mixed paint-rock, gravel and small boulders, a large 5-foot boulder; sand and gravel.

A flood-gate was built in the drift, 50 feet from the raise, to stop any possible sand runs. The sand piled up 4 feet high against the gate. The sand is very fine and resembles clay; it appears to be impervious to water. A layer of this fine, clay-like sand (50 feet thick at the shaft) overlies the ore formation and separates the ore and capping from the overlying quicksand. It is this layer that prevents the draining of the surface. The raise proved to be dry and practically no water came down with the sand. This was considered a favorable condition from the slicing standpoint.

A drift was started late in May in the neighborhood of the test raise, in order to open a slice, with a view to caving as large an area as possible.

On June 12th the total development aggregated 12,061 feet and slicing was begun. Drifting from now on was to be practically confined to the sub-level. The weekly progress in the slices ranged from 150 to 300 feet and on August 17th it aggregated 1,800 feet. On that day a stope on No. 8 sub, 200 feet long and 100 feet wide was caved. The stope filled with boulders and gravel. No sand or water to be seen.

On September 30th the aggregate of drifts on the sub was 9,922 feet (total for the mine, 15,047 feet), and the aggregate of slice drift was 4,106 feet. A very successful cave is reported about this time on No. 5 sub. An area 150 feet long by a width varying from 50 to 150 feet caved, and made two holes through to the surface. The stope filled with mud that was not quite wet enough to run.

During the week ending October 21st, four days were lost by a strike. The company had distributed a number of booklets setting forth "Rules for the Prevention of Accidents." These were written in English and readily accepted and receipted for by the miners who spoke some English with the exception of the Finns, who were

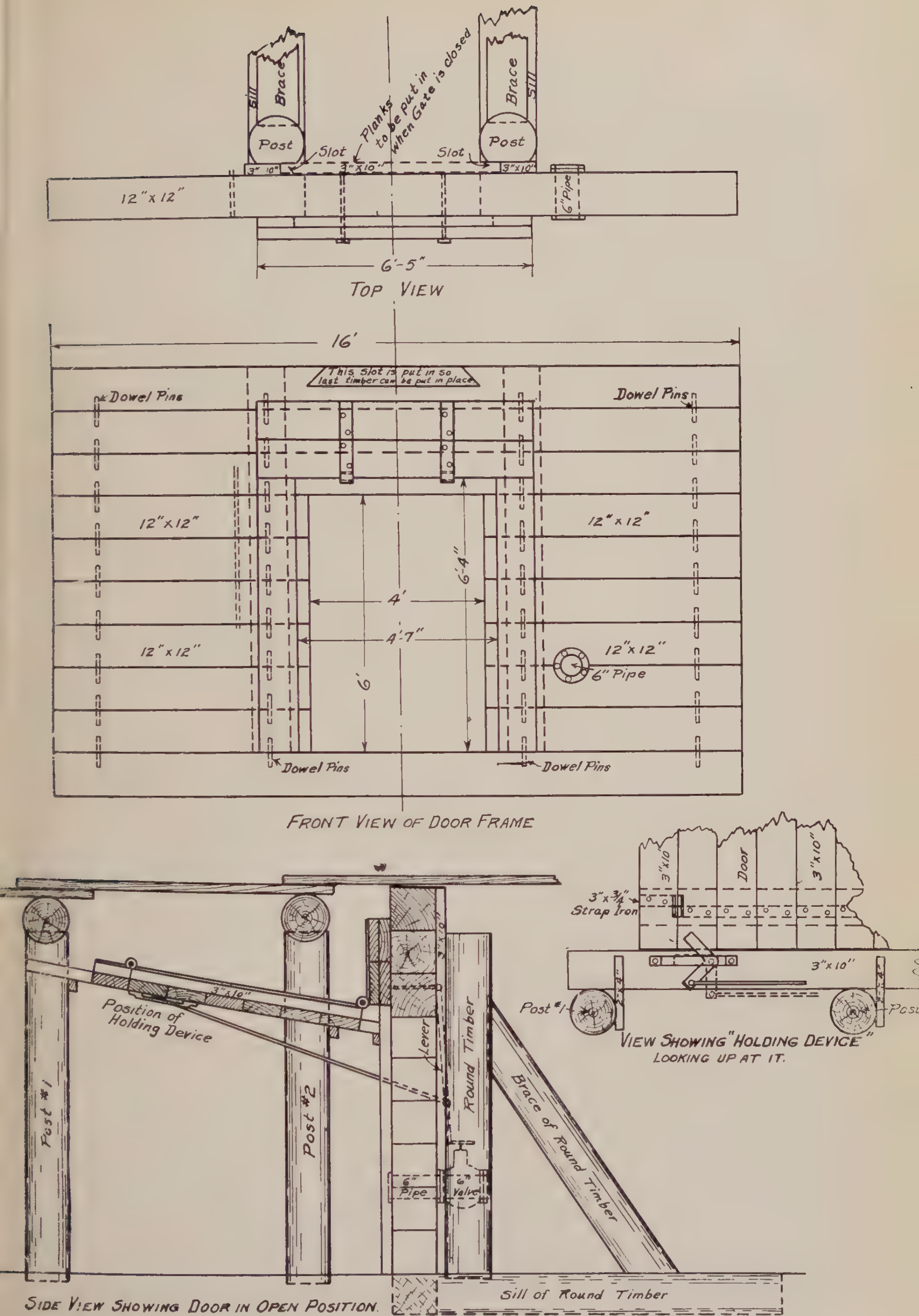


FIG. 54. Bangor Flood-Gates.

suspicious. A personal injury suit brought by a Finn had recently been won by the company, and there was a strong feeling among the Finns, who were decidedly inclined toward Socialism. Early in October the company tried to introduce a special application card among the men. The latter feared that it was a "black-list" or that it would operate to their disadvantage in the case of injury, and refused to sign. The company did not press this, but did insist on the men accepting and receipting for the instruction book. The matter was carefully presented to the Finns from the personal safeguarding point. One of them, on refusing the book, was discharged. He called out the other Finns, and they by threats and violence called out the other nationalities. A couple of days later the agitators had been convinced by outside counsel that the books were harmless and before the week was over all hands resumed work. The incident is illustrative of the ignorance and superstition that characterize the great bulk of labor available on the ranges.

On October 28th the advance on the sub-level aggregated 10,369 feet of drifts and 4,863 feet of slicing. One of the unexpected and disappointing incidents of that week was a quicksand run in a slice on No. 8 sub. An unsuccessful attempt was made to stop it with improvised barricades. The sand finally drove the men back to the flood gates, filling about 150 feet of drift. This cave was a surprise since during the summer months several successful caves had been brought in which gave no trouble from sand or water; and it was thought that the mine had safely passed through the most critical stages. On November 11th drifting was resumed on the main-level and the water immediately increased to 4,100 gallons.

On November 14th, at 5:30 p. m. the stope at the west end of No. 5 sub began to cave. All approaches to this stope were guarded by flood gates or timber pillars that would have stopped an ordinary cave or mixed run such as the mine had experienced hitherto. With the breaking of the taconite capping in the back of the stope a great flood of water came down carrying a small quantity of sand. This water ran through the timber pillars and out into the working drifts of No. 5 and No. 6 subs. The water was about three feet deep on the main traveling drift on No. 5 sub and coming rapidly toward the raises.

Usually when ground seems dangerous and a cave is feared, a "safety-man" is stationed at the flood gates and given general charge of the district. It would be his duty to see that all hands are out and the gates properly closed in the event of a rush. The bottom 12 inches of the gate is hinged and the gate may be closed with a foot or more of mud on the floor. It is planned to make future gates with an opening or window near the top and a hinged flap; this would permit a man to pass through a closed gate with 2 or 3 feet of sand behind it. In this case there was no safety-man at the gate and the last man going through was going too fast to think of stopping and pulling the trigger. On former occasions when a cave occurred a thick mud would flow slowly; this time the men were panic stricken by the sudden rush of water. The mine captain and a few picked men went down at once and succeeded in partially shutting the gate perhaps 30 minutes after the cave occurred. The flow through the open space on the bottom under the partially closed gate was stopped by ramming hay

and plank under the hinged bottom. The details of these gates are shown in Fig. 54.

During the week several attempts were made to re-open the caved ground, but in each case the men were driven out by new rushes. A hole 100 feet in diameter and 50 feet deep appeared on the surface over the stope, just like a milling pit. This cave temporarily lost 9 working places and will materially affect production for some time to come. No. 6 sub-level was cut off from No. 5 sub by a strong barricade and much sand was removed. Wednesday's night shift closed all the gates on No. 5, 6, and 8 sub-levels to prevent a possible sand run during the next 24 hours (holiday). On Friday morning all the drifts on the south side of No. 5 sub were found full of sand. This drove out the four remaining gangs on No. 5 sub. During the 6 weeks ending December 30th the total progress was: drifting, 536 feet; raising, 104 feet; slicing, 751 feet. During this period the mine was shut down for 1½ days owing to a defective throttle on the hoisting engine. On January 1, 1912, the advance on the main level aggregated 5,285 feet and on sub-levels 11,143 feet of drifting and 6,060 feet of slicing.

During the months of July, August, and September, 1911, the monthly tonnage was 14,000 to 15,000 tons mined from 14 to 15 working places. As the mine opens the number of working places will increase to 25 normally with a possible maximum of 35. The normal full monthly tonnage for which the mine is planned is 30,000. At present the slicing practice varies a little from the standard on account of peculiar conditions. The initial caving of a tough taconite capping always presents some difficulty; it often is slow, expensive work. The caving frequently does not readily follow. When the capping does break, it may come suddenly, settle on the timbers and break them, endangering the lives of the men in the slice. Single width drifts are run 8 feet high and enlarged to 15 feet high with a double deck drift-set. The drifts are run in the solid, carrying a 10-foot pillar between the drift and the cave. When the drift is completed, the pillar is mined out on the retreating plan by a series of short slices at right angles to the drift. The miner is thus safeguarded against a sudden rush from caving capping and from sand rushes. The slice drifts have their own sand gates and there are emergency gates in the main travel-drifts on each sub.

This preliminary slicing is slow and expensive, about one-third of the ore is mined from the solid. After a drift is finished and the 10-foot pillar between it and the cave is sliced back, another drift is opened so as to leave a 10-foot pillar between it and the cave. A number of gangs are cleaning up sand and doing other special work; this is all charged to mining and materially pulls down the average per man.

During the month of September of this year 14,000 tons were mined.

The tonnage per man, including every man on the job, was 2.13.

The tonnage per man underground, excluding the men on development work, was 5.2.

The tonnage per man slicing, determined from the performance of 10 gangs of 4 men (2 day and 2 night shifts), who, in 26 working days, mined 8,480 tons, is 8.15 tons.

Standard drifts in this mine are 8½ feet by 9 feet high. They are run by a 4-man

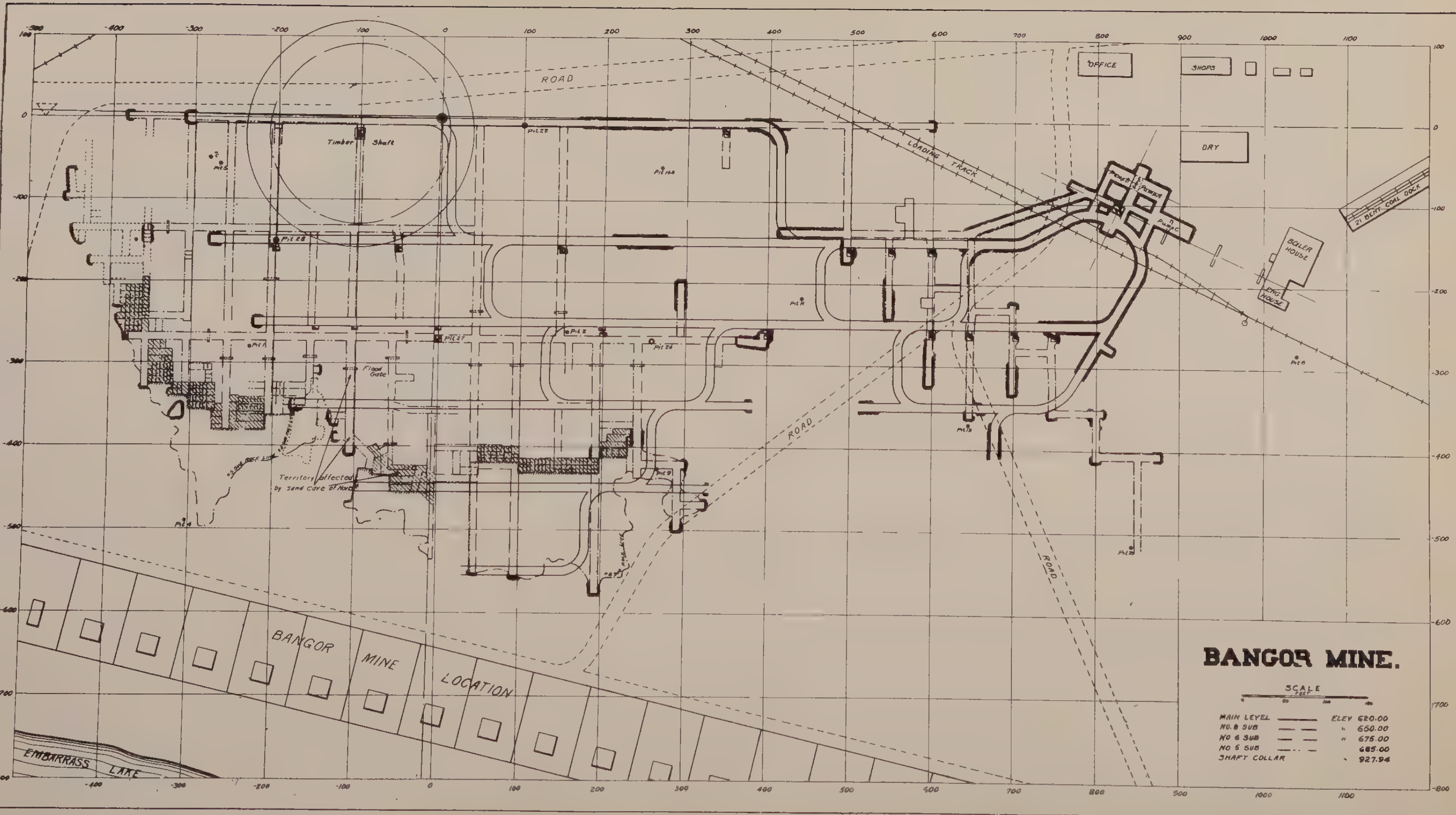


FIG. 55. Map showing Bangor Develo^rment to Jan. 1, 1912.

gang, 2 men on each of 2 10-hour shifts. The ground is too hard for auger or hand work and a 2½-inch Rand drill (weight 135 pounds) is used. The drilling time is short, usually 2½ to 3 hours. The advance is on an average 25 feet per week, working 2 shifts 6 days.

The following table gives the labor distribution for this month.

ANALYZED LABOR STATEMENT BANGOR MINE

For the month of September, 1911

	Av. men	Days worked
Office	1.8	47.80
Mining engineers	0.7	18.25
Surface foreman	...	....
Master mechanic	1.0	26.00
Cutting lagging (contract)	14.3	371.00
Engineers	2.0	60.00
Firemen	4.0	116.00
Pipemen	1.5	40.00
Blacksmiths	0.8	20.50
Helpers	1.1	28.75
Carpenter foreman	1.0	28.00
Helper	0.1	2.00
Teamsters	1.0	26.00
Dry, barn and janitor	2.4	61.50
Surface laborers	3.4	88.25
Surface timbermen	2.0	52.00
Mason	0.3	8.00
Timber framers	1.6	41.75
Steam shovel operators	3.0	85.00
Laborers	2.1	53.50
Test pitters	0.2	4.25
Coal handler	1.0	30.00
Ore sampler	0.5	12.25
Watchman	0.2	6.00
Boiler cleaner	0.1	1.00
Total Surface	48.1	1279.80

	Av. men	Days worked
Mining captain	1.0	26.00
Foreman	2.0	52.00
Miners, day work	1.8	46.00
Miners, contract	79.0	2053.50
Safety men	3.1	79.50
Pipemen	0.4	10.00
Timber foreman	0.9	22.50
Timber helpers	5.0	130.00
Chute men	1.9	49.50
Mule drivers	4.0	104.00
Skip tenders	2.0	52.00
Pumpmen	4.0	120.00
Powder men	2.0	51.00
Track men	1.9	48.50
Laborers	3.0	77.00
Total underground	112.0	2921.50
Superintendent and assistant	1.1	28.60
Total Surface and Underground	161.1	4229.90

Summary Bangor development.—The shaft was started by the Foundation Company in the fall of 1906. The excavation was advanced sufficiently to permit building the shoe. Upon completion of the shoe, work was stopped.

Sinking was resumed in May, 1907, and the shaft was sealed in the rock at a depth of 134 feet and turned over to the Mining Company in October, 1907.

The shaft was sunk by the Mining Company to a depth of 314 feet; the station was opened on the 308-foot level and drifting was begun the end of November, 1908.

On June 12, 1911, slicing was begun and the total development to that date aggregated 12,061 feet.

Total number of man-shifts worked, 86,987.

Pumps handling 4,000 gallons per minute.

On January 1, 1912, total development aggregated 16,428 feet. Fig. 55 shows a plan of the Bangor development to this date.

Total development tonnage from drifts and raises, 110,000 tons.

Total slicing footage, 6,060 feet, yielding 56,728 tons.

Total man-shifts worked to date, 113,734.

Production, 1909, 3,837 tons
Production, 1910, 46,313 tons
Production, 1911, 116,578 tons

Total production, 166,728 tons

The following table gives the labor statement by months to January 1, 1912:

MONTHLY LABOR STATEMENT BANGOR MINE

DATE		SURFACE LABOR			UNDERGROUND LABOR			TOTAL LABOR	
Year	Month of	Men per 24 hours	Days' work per month	Shaft-men per 24 hours	Miners per 24 hours	Total men under-ground	Days' work per month	Men per 24 hours	Days' work per month
	Oct.	14.5	375.25	9.0	2.6	11.6	303.25	26.1	679.00
1907	Nov.	10.0	266.75	7.0	1.6	8.6	233.75	18.6	500.50
	Dec.	15.8	411.00	10.9	4.7	15.6	405.75	31.4	816.75
1908	Jan.	17.5	456.5	10.7	7.5	18.2	472.50	35.7	929.00
	Feb.	21.8	568.25	18.0	4.9	22.9	595.50	44.7	1163.75
	March	20.0	563.00	20.1	4.0	24.1	645.25	44.1	1208.25
	April	22.6	589.50	16.7	5.3	22.0	558.25	44.6	1147.75
	May	30.9	802.50	19.5	6.3	25.8	679.25	56.5	1481.75
	June	39.3	1022.75	3.5	6.0	9.5	250.75	48.8	1273.50
	July	30.3	797.75	18.8	7.7	26.5	698.00	58.8	1495.75
	Aug.	28.4	757.00	19.6	8.4	28.0	728.00	56.4	1485.00
	Sept.	25.6	707.00	16.6	7.9	24.5	642.75	50.1	1349.75
	Oct.	33.9	918.75	14.5	8.0	22.5	615.50	56.4	1534.25
	Nov.	36.9	973.75	13.5	9.2	22.7	524.25	59.5	1548.00
	Dec.	44.3	1135.75	19.2	12.9	32.1	812.25	76.4	1948.00
Average for year 1908		29.29	774.37	15.89	7.34	22.23	606.02	52.59	1380.39
1909	Jan.	34.8	942.7	....	18.2	28.1	863.0	62.9	1805.71
	Feb.	36.8	902.71	....	20.0	35.0	851.5	71.8	1754.21
	March	32.5	901.12	....	17.8	32.9	901.8	65.4	1802.91
	April	33.0	936.16	....	12.7	26.7	700.5	59.4	1636.66
	May	29.0	867.35	....	8.6	19.1	592.75	48.1	1460.10
	June	27.6	743.00	....	10.0	19.1	499.50	46.7	1242.50
	July	27.1	745.25	3.5	16.8	38.2	915.00	65.3	1660.25
	Aug.	33.2	881.75	16.4	18.3	48.1	1275.25	81.3	2157.00
	Sept.	34.8	908.75	13.9	25.5	55.0	1443.00	89.8	2351.75
	Oct.	37.4	1019.50	11.0	32.8	62.0	1662.00	99.4	2681.50
	Nov.	36.3	974.50	....	52.9	74.8	1982.25	111.1	2956.75
	Dec.	40.9	1103.75	....	42.1	67.4	1775.75	108.3	2879.50
Average for year 1909		33.61	910.54	....	22.97	42.20	1121.85	75.79	2032.40
1910	Jan.	43.2	1142.00	....	43.9	68.4	1742.00	111.6	2884.6
	Feb.	58.7	966.75	....	12.3	26.9	670.00	65.6	1636.75
	March	36.8	1009.50	....	40.9	65.0	1712.5	101.6	2722.00
	April	34.4	921.75	....	47.3	76.0	2002.25	110.4	2924.00
	May	31.9	837.50	....	34.3	59.5	1545.25	91.4	2382.75
	June	22.0	599.75	....	26.2	48.5	1269.00	70.5	1868.75
	July	31.9	829.45	....	27.6	50.8	1279.00	82.7	2108.45
	Aug.	26.7	743.55	....	27.1	48.9	1346.25	75.6	2089.80
	Sept.	26.4	713.30	....	28.6	52.2	1373.5	78.6	2086.80
	Oct.	32.0	859.05	....	34.5	59.0	1535.0	91.0	2394.05
	Nov.	30.1	802.80	....	51.1	78.2	1978.0	108.3	2780.80
	Dec.	34.7	940.80	....	54.8	81.5	2139.75	116.2	3080.55
Average for year 1910		34.06	863.85	....	35.71	59.57	1549.37	92.37	2413.35
1911	Jan.	37.0	951.5	....	62.1	91.3	2310.00	128.3	3261.50
	Feb.	35.9	899.75	....	66.6	94.8	2290.25	130.7	3190.00
	March	30.3	836.00	....	55.0	82.6	2247.5	112.9	3083.50
	April	31.3	799.50	....	51.1	76.6	1863.75	107.9	2663.25
	May	28.4	786.55	....	54.4	80.6	2192.75	109.0	2979.30
	June	29.4	793.55	....	84.6	112.7	2944.00	142.1	3737.50
	July	35.2	916.80	....	75.4	109.2	2751.00	144.4	3667.80
	Aug.	46.0	1265.30	....	81.1	115.6	3134.00	161.6	4399.80
	Sept.	48.1	1279.80	....	80.8	112.0	2921.50	160.1	4201.30
	Oct.	43.2	1148.30	....	66.2	93.4	2426.00	136.6	3574.30
	Nov.	39.8	1028.55	....	86.2	124.8	3121.25	164.60	4149.80
	Dec.	33.5	856.30	....	51.5	89.1	2160.50	122.60	3016.80
Average for year 1911		36.50	963.49	....	67.91	....	2530.20	135.06	3493.73
Average three years, 1909-1911		33.36	878.06	....	32.98	55.51	1451.86	88.95	2329.96

UNDERGROUND MISSABE MINING COSTS

The cost of equipping and opening an underground mine on the Missabe varies with the location; the tonnage developed; the operating conditions, especially the difficulties encountered from sand, water, quicksand, broken up or mixed ore; and, finally, the condition of the ore-market during the development period. The extreme variation to which these conditions are subject makes it somewhat difficult to make serviceable general statements. An important factor in the initial outlay is the size and completeness of the mine plant and the grade of the equipment. This is largely regulated by the total tonnage disclosed by drill prospecting. On a large tonnage the ultimate net saving resulting from an efficient equipment will, of course, warrant incurring the extra cost of such equipment. A small tonnage, on the other hand, might yield a very satisfactory profit on a small working capital, which would compel a minimum plant investment, lacking probably in many details considered essential to economical operation at a large mine. The following estimates include the entire cost of opening, including surface plant complete, shaft sinking, underground haulage and pumping equipment, opening main level and sub-levels to a point where a reasonable slicing production may be maintained:

A small mine, close enough to a center of population to eliminate the housing problem, with a shaft not over 250 feet deep, pumping not over 250 gallons of water per minute during sinking and 500 gallons during development, having no operating difficulties, with an output of not over 500 tons a day, and with a life of not over 15 years, could probably be equipped and opened on a cash capital of $50,000, provided the market could readily absorb the development ore. Under favorable conditions, the lateral development will more than pay its way after the ore has been reached. With comparatively poor ore, unfavorable operating and market conditions, the operating capital might have to be doubled or even trebled. The average Missabe mine planned for a daily output of 1,000 to 1,200 tons, presents no serious difficulties and $150,000 to $200,000 will usually open and satisfactorily equip it.

Many properties have expensive sinking problems and subsequent heavy station pump installations to face and may be called upon to spend from $200,000 to $300,000 in surface equipment, shaft sinking, and pump equipment alone before any lateral development is begun. If to this be added much development work in rock or in low-grade ore that has to be stock-piled on account of the unfavorable market conditions, then the cash capital invested may easily reach $400,000 to $500,000 before returns begin to come in. The experience at the Whiteside Mine serves as a concrete illustration of this point. Development began in March, 1909, and ore production from development work began in March, 1910. A good grade of ore would have been imme-

diately absorbed. During the 1910 shipping season operations would have been credited with $75,000 or more from development ore sales, representing about 25 per cent of the gross expenditures incurred to the end of the 1910 shipping season. Similarly the entire production to date of 250,000 tons would have been absorbed. Operations would have been credited with $375,000, assuming the ore to be marketable at its value—practically 75 per cent of the total gross expenditures to January 1, 1912.

The operating company in this case had the advantage of being able to force a market for its mine product up to the capacity of its furnace plant to absorb ore of this grade. Even under these favorable conditions the credit for ore sales is under $200,000—less than 40 per cent of the gross expenditures to January 1, 1912. The development ore mined to the end of April, 1911, is still on the stock pile. The plight of a mining company with no smelting affiliations, under the market conditions that have obtained for the past three years, may be imagined.

The Bangor Mine was cited as an illustration of the most difficult conditions so far found on the Missabe. The shaft was sunk 135 feet under air-pressure and 209 feet additional in rock, 2,000 gallons of water being pumped per minute. The aggregate cost of shaft, stations, pump station, and underground pump equipment must be at least $200,000. This is exclusive of surface plant. Thereafter development work was prosecuted for 3 years with an average crew of 90 men, the pumps handling from 3,500 to 4,000 gallons per minute. The heavy flow of water increased noticeably each time a new breast was opened. As a precaution against flooding, development was restricted to a few headings at a time, and progress was much slower than is customary, the overhead charges per foot of development work thus running unusually high. The pumping cost alone must have averaged $5,000 per month and the total yearly operating expenses must have exceeded $150,000. A company operating under such difficulties will very likely approach a gross outlay of a million dollars in plant, equipment, operating, and interest charges before the mine is fully developed. From half to two-thirds of this would likely be spent before appreciable returns begin to come in.

Cost of underground ore.—A study of Missabe underground mines brings out strongly the fact that there is much variation in operating conditions and a surprising fluctuation in the cost of ore. The average cost of ore in a district is no criterion of the cost at any particular mine. This is one of the principal reasons assigned by the operators for their refusal to publish mining costs. A bare statement of the cost of ore might easily be misconstrued. The average Missabe mine is not operated as a unit, but as one of a group. The grade of ore in one mine is frequently complemental to that of another.

The operators demand certain grades and tonnages from the group as a whole. Economical operating conditions at one or more mines of the group may have to be sacrificed to satisfy business requirements from the smelting and transporting standpoint. Thus a property might be opened and equipped to produce ore at a low cost when operating at a normal production of 1,000 to 1,200 tons daily. The output might, for business reasons, be curtailed to half that amount. There would be a

corresponding reduction in the men employed in actual stoping and the effect on the stoping cost proper might be inappreciable. The other operating expenses such as pumping, hoisting, company tramming, surface expenses, and the overhead expenses would in the aggregate show very little decrease. This element in the cost of production (from 40 to 60 per cent of the total) would therefore practically be doubled. The cost of ore during the development period, during the decline of the mine, and during any period when production is below normal, will therefore be decidedly in excess of the normal production cost.

Stoping or slicing contracts are let either per foot of advance, one set wide or one set high (per square set in square-set slicing), or per car of a certain measurement. Cars used in different mines vary in size. Moreover, there is a great variation in the porosity and composition of the ores in the different districts in adjoining mines or even in the same mine. In the Chisholm district a series of tests on ores of varying composition show an extreme variation of 40 per cent in the cubic contents per ton of ore. Therefore, the same contract price per foot, per set, or per car in different mines might give widely varying prices per ton. Operating conditions such as regularity of floor and roof, rock intrusions, hardness of ore, weight thrown on timbers, water and sand difficulties, regularity of output owing to variable demand, are all subject to change. Therefore the cost of slicing varies between wide limits in different mines and even in parts of the same mine. This variation is quite independent of the variations before mentioned as affecting the total cost of ore. A reduction in the tonnage would not seriously affect the cost of ore in the trammers' chutes underground. Contract prices are set by the mine captain, who must give careful consideration to such questions as thickness of ore, hardness, intrusions of rock, length of tram and number of transfers.

For a 56-cubic-foot car containing perhaps an average of 3 long tons of ore, the contract price would vary from 65 cents to 85 cents per car, and it may run up to $1. This includes mining-labor, timbering-labor, mine supplies, and tramming to the chute (150 to 400 feet), whence it is drawn by the company trammers. Contracts put out all the way from 10 to 12 cars per shift in hard places to 15 to 18 cars under normal conditions in drift-slicing and from 25 to 35 cars from the upper sets in square-set slices. Slicing contracts per foot run from $2 to $3. In one mine, under fair working conditions, good ore, in a double slice, height 14 feet (12½-foot post), with a 400-foot tram, the men were paid $2.50 per foot, single slice (7 feet) width. Two men got out from 14 to 16 cars in a ten-hour shift, besides setting their timber. In a neighboring mine under similar conditions with a tram of 150 feet, the price was $2.25. Another contract in this mine, running a 10½-foot slice (9-foot post), paid $2.75 per foot; in this case there was a 400-foot tram with 5 transfers to get the ore into the chutes.

On square set work the contracts usually run from 5 to 10 cents lower than on drift-slice work because shoveling is greatly reduced. These prices apply to the Hibbing and Chisholm districts for the years 1909-1911. One mine in the Chisholm

COST SHEET ________ IRON MINING COMPANY

	Month of December, 1911 Product 22,800 Tons					From January 1, 1911, to December 31, 1911 Product 250,000 Tons				
	Labor		Supplies		Total Cost Per Ton	Labor		Supplies		Total Cost Per Ton
	Total	Per Ton	Total	Per Ton		Total	Per Ton	Total	Per Ton	
Mining Expense										
Mining	$9,038.74	.396	$1,289.59	.057	.453	$91,484.49	.366	$15,289.46	.061	.427
Timbering	1,497.93	.066	2,239.71	.098	.164	14,615.53	.058	24,270.10	.097	.155
Tramming	883.86	.039	184.02	.008	.047	9,198.65	.037	1,778.88	.007	.044
Pumping	298.63	.013	458.97	.020	.033	3,307.69	.013	3,633.13	.015	.028
Underground tracks	113.95	.005	93.11	.004	.009	1,283.01	.005	364.72	.001	.006
Mining captains, shift bosses and time keepers	488.70	.021	....	....	.021	5,629.63	.023	....	....	.023
Sinking and repairing shafts	25.32	.001	23.65	.001	.002	446.50	.002	382.85	.001	.003
Mine exploration	281.30	.012	31.54	.002	.014	994.16	.004	103.19	.001	.005
General underground expense	269.05	.012	....	....	.012	3,126.33	.012	....	....	.012
Total mining expense	$12,894.48	.565	$4,320.59	.190	.755	$130,085.99	.520	$45,822.33	.183	.703
Surface Expense										
Stocking and loading stock pile	$335.81	.015	$45.87	.002	.017	$2,821.41	.011	$871.38	.004	.015
Pocket loading	....	....	....	....	....	474.73	.002	.70	....	.002
Hoisting	129.30	.005	284.08	.013	.018	1,251.66	.005	1,693.90	.007	.012
Building repairs	....	....	5.38	....	....	213.46	.001	781.47	.003	.004
Machinery repairs	230.92	.010	615.92	.027	.037	2,383.77	.010	2,295.28	.009	.019
General surface expense	129.18	.006	22.56	.001	.007	1,832.15	.007	71.00	....	.007
Miscellaneous	24.68	.001	....	....	.001	500.95	.002	....	....	.002
General office expense	108.55	.005	80.62	.003	.008	1,288.94	.005	925.49	.004	.009
Total surface expense	$958.44	.042	$1,054.43	.046	.088	$10,767.07	.043	$6,639.22	.027	.070
Total Cost of Mining	$13,852.92	.607	$5,375.02	.236	.843	$140,853.06	.563	$52,461.55	.210	.773

district during the year 1911 established a flat contract rate of 30 cents per ton. The net labor cost per ton slicing at this mine varies from 23¼ to 24 cents.

The stoping cost of ore, i. e., cost of ore delivered by slice contractors to their local tramming pockets, varies with circumstances from 40 to 50 cents per ton during normal activity of the mine. Development ore and ore drawn from old pillars prior to final abandonment is not included in this figure. This cost includes contract labor properly chargeable to slicing, cost of timber, cost of explosives and other supplies used in the slices. It may be subdivided roughly into: Mining and local tramming, 55 to 60 per cent; timbering (labor and supplies), 40 to 45 per cent. The total cost of producing, i. e., the cost on cars, exclusive of overhead expense, is subject to greater variation, depending upon the relation of fixed and operating charges to the varying tonnage, the pumping problem, and other difficulties. It may be taken as ranging from 65 to 85 cents for the normal case; and 90 cents to $1.10 for very wet mines operating under great difficulties or for the smaller mines having a heavy fixed charge and a comparatively small tonnage. These costs include depreciation and construction account extinguishment, charged off against the total tonnage; they do not include head-office administration, taxes, royalty, and kindred charges.

The table (p. 128) gives the cost of ore for the year 1911 and for the month of December, 1911, at a well-managed underground mine on the Eastern Missabe. The operating conditions at this property are fairly good. It has been a continuous shipper for a number of years and the total shipments to date are a trifle under two million tons.

At one property, developed within the past few years under operating conditions that are by no means ideal, the ore cost-sheet for the first 12 months of development work varied downward from $4 to $1.25 per ton, the cost averaging $1.824 per ton for the first 75,000 tons. All operating costs are charged to the ore; equipment is not charged to operating account but to a construction and equipment account, charged off monthly against the ore as a fixed "plant extinguishment" charge per ton mined. The cost for the second year was reduced to $0.872 per ton for some 175,000 tons, about 60 per cent of which was slice ore and 40 per cent development ore. The average cost for the two years was $1.144 per ton. Since then the cost has been reduced to from 85 to 75 cents per ton inclusive of all charges except plant extinguishment, royalty, eastern office expenses, and similar charges. Cost sheet for two months at this property follows. The variation in cost is in part due to differing proportions of drifting and slicing ore.

	First month	Second month
STOPING COST—	Cents	Cents
Contract and company account labor..............	26.3	34.6
Powder	4.1	8.0
Other expenses	1.9	0.8
	32.3	43.4

TIMBERING—		
Timber, all timber used in mine. Timbering labor exclusive of timbering in stopes which is part of general contract	8.8	7.5
TRAMMING—		
All company account tramming exclusive of contractors' delivery to primary chutes which is part of stoping contract	7.7	6.8
PUMPING—		
Labor, power, and supplies	4.2	5.8
LIGHTING	0.4	0.4
HOISTING—		
Labor, power, supplies, and repairs	3.3	5.4
GENERAL MINE EXPENSE—		
Superintendence, dry house, laboratory, engineer, direct office expense	7.5	8.6
TOTAL MINING COST	64.2	77.9
STOCK PILE OR LOADING	0.4	0.2
GENERAL EXPENSE—		
Taxes, insurance, workmen's fund, depreciation	7.6	9.5
	72.2	87.6
Tonnage mined	21,500 tons	18,600 tons
Percentage slicing ore	62.5%	35.0%
Percentage drifting ore	37.5%	65.0%

OPEN-PIT MINING

Opening a Missabe pit is largely a problem in civil and mechanical engineering, involving the study of the following questions for each case:

1. Proportion of overburden to ore, taking into account both the relation of total volume of stripping to total ore tonnage and the relative thickness (depth) of overburden and thickness (depth) of ore. The admissible ratios are stated on page 45. Increased wages, decreasing efficiency of labor, increased timber and supply costs, together with the enormous tonnages called for (often 4 to 5 times the possible output of a fully opened underground mine) all make for an extension of the open-pit method and an increase in the allowable ratio of overburden to ore.
2. Location of the best approach for the tracks, considering both the stripping and subsequent ore transportation.
3. Stripping dumps.
4. Track system.
5. Drainage.
6. Mechanical equipment.

LAYING OUT THE PIT

The engineer makes special maps showing the following contours: Top of ore; bottom of ore; top of high-grade ore, etc. The cross-sections are supplemented by such longitudinal sections as are necessary to show clearly the position of the various layers of ore and rock intrusions. After all the drill holes have been carefully examined and it is definitely decided what portions shall be mined by underground methods or "scrammed," a line is drawn around the open-pit drill holes a reasonable distance away from them according to the open-pit "strength." This line is called the "crest of ore" in the proposed pit. Another line is drawn 20 feet outside of the first line and called the "toe of stripping." Allowing a 1 to 1 slope for the stripping bank the "crest of stripping" is next located. The proposed pit must be fairly regular in outline. The approach must be carefully considered from both the stripping and the mining standpoint. Often the best solution is to cut a temporary approach for the stripping tracks, especially if a long haul to the dump can thereby be avoided.

When estimating the ore tonnage available to steam shovels from the proposed pit, a 1½ to 1 slope is used from the crest of ore to the bottom of the pit, on all sides of the pit but one. The side that is to carry the track system is figured with a 1 to 1 slope.

The selection of stripping limits is further governed by the fact that it is desirable to have comparatively straight lines rather than undulating lines. The thickness of an ore-body on the edges is apt to be very irregular and the drill holes on

the edges are likely to show alternating stripping and non-stripping holes. The stripping limits are therefore not laid out to the edges as the resulting lines would be very undulating, something to be avoided in so far as possible. Test-pits are put down at regular intervals on the rear of the shovel on its last cut along the stripping limit laid out. If it develops that a ton of ore may be uncovered per yard of stripping by taking another cut with the stripping shovel, this may be readily done. In this way stripping beyond the economic limit is avoided. Any remaining ore is reached by drifting in from the pit and mining by ordinary underground methods. The ore is trammed out and stock piled in the pit whence it is loaded with steam shovel into ore trains. This is referred to as "cleaning up the edges."

Stripping banks.—Stripping banks generally have, as before stated, a 1 to 1 slope and a 20-foot berm. When the pits are deep and the banks must stand for many seasons, these slopes should be broken at least every 75 feet and preferably every 50 feet, with a berm of 25 to 30 feet. It is the general practice to leave about a 10-foot cut for the "clean-up" shovel, a smaller shovel having 6 to 8 pit men working in conjunction with it. The remainder is cleaned off "by hand," i. e., team and scraper work, pick and shovel and wheelbarrow work, etc. (see Fig. 95). Considered singly this is quite an item of expense; divided into the total tonnage or even into the tonnage of the first cut it is negligible and it is the only way of cleaning thoroughly down to the top of the ore and insuring a clean top layer. The tops of the ore-bodies are by no means smooth and uniform. A great many knolls and hollows occur, making a very uneven top. Drill holes may miss these entirely. In one property just stripped several large hollows from 20 to 40 feet deep were visible. Lining in the drill holes from the coördinate pegs on the banks of the pit showed that out of several depressions only one was hit by drill holes spaced at 300-foot intervals; while 200-foot drill holes would have disclosed most of the depressions. This in itself is not a very serious matter, though it adds appreciably to the cost of stripping and cleaning the tops and entails a certain amount of irregularity. A stripping plane or level must be established and the knobs above this are mined and stock-piled as stripping progresses. The ore shovels may also be blocked up with ties laid crib fashion when going over cleaned or stripped depressions between knobs of ore. When the layer of ore left for the "clean-up" shovel is so thin that there is not enough sand to keep a locomotive busy, the shovel frequently "casts up," i. e., banks the sand on top of the next cut to one side, whence it is removed as a part of that cut. After the ore-body is stripped, the upper portion is carefully test-pitted to a depth of 40 to 60 feet. This work is for the purpose of verifying the drill sampling and affords an opportunity to fill out any incomplete information. It gives the operating department a basis for grading, classification into layers, and intelligent direction of steam shovel work in ore.

Stripping dumps.—In the early days of Missabe mining very little care was exercised in the matter of selecting non-ore-bearing ground for dumping purposes and several good stripping propositions were ruined or badly handicapped by dumping from 20 to 40 feet of stripping upon them. At present dumps are confined to barren

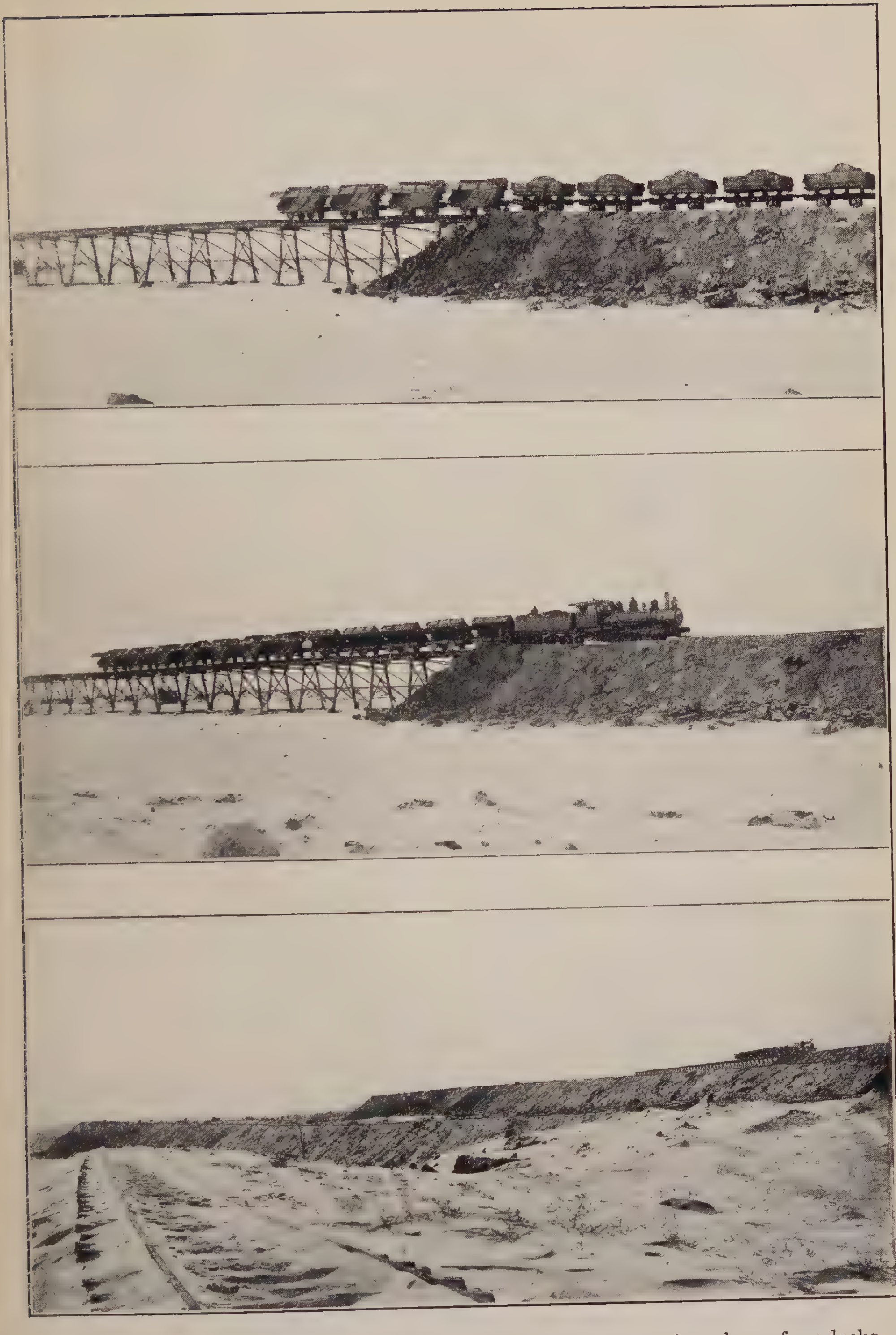


FIG. 56. Stripping Train Dumping from Trestle. The lowest view shows four decks of the dump.

ground or to ground where the ore is overlaid by so much surface that stripping is manifestly out of the question, even should the present ratio of permissible volume and depth of over-burden to ore be materially increased. Of course, underground property is in no wise affected by having its surface covered with a stripping dump. The mining companies now test all ground offered for dumping purposes by putting down five drill holes to a "40."

A low piece of ground sloping away from the initial dumping point and giving a down-hill haul from the pit is ideal. A dump is started by backing the train out of the pit (locomotive in the rear pushing train out), and dumping the train load beside the track, alternate cars on opposite sides. The track is then jacked up and this process is continued until the dump has reached the required height. After this the dump is "fanned out," i. e., extended by throwing track sidewise 4 feet, thereby building up the dump. This is the old method, which has some points in its favor and is advocated by many. The more general method, however, is to build a trestle. These trestles are of light construction, 20 to 25 feet high. They are built strong enough to carry the *empty*, but not the *loaded*, train. A trestle of desired length and height is built and filled in as follows (see Fig. 56):

A loaded train is pushed out of the pit and stopped when the first car reaches the trestle. After this car is dumped beside the track, the train is moved up one car length (the empty car being run out on the trestle), and the next car is dumped on the opposite side of the track. This is kept up until the whole train is dumped and all the empties are on the trestle, the engine only being on firm ground. The dumping crew shovel to the center and jack up the track which is packed down by the next loaded train running over it. Trestle dumps give a great deal of trouble. The cars dump well away from the rail and the dirt falls unequally around the legs of the trestle. The cost of a well-built 25-foot single-track trestle of this type ranges from $2.50 to $4 per lineal foot, depending upon the weight of equipment it is to carry. When the trestle is established and filled in, the track is thrown and the dump is fanned out. A dump may be established on either or both sides of the main track, depending on the topography and general conditions. Often the trestle is several thousand feet long and several dumps are started at intervals along its length. The desirable length of a dump ranges from 1,200 to 1,400 feet, the desirable height from 20 to 40 feet, the latter being considered ideal. Sixty-foot dumps are excessive, there is too much settling of the dump and consequently lowering of track. High dumps are also apt to shelve off. If the dumps are soft, the outer rail is sufficiently elevated to allow for any settling or pounding occurring as trains dump. Straight dumps are best, though it is difficult on the straight dump to "keep up the end," i. e., maintain the length of the dump. Many contractors, therefore, maintain a straight dump curved out on the extreme outer ends.

Theoretically, circular dumps, i. e., kite-shaped tracks, would seem to be ideal, but in practice they prove very troublesome. The track binds and is hard to throw as the dump extends. Straight tracks on the other hand are easy to throw. One property reports that in heavy work (bad weather and sticky dirt) a crew of 16 men

easily took care of a straight track, while 40 men were required to throw a curved track. Rails are laid with joints open and on the first throw the crew begin at the end, taking up slack and working back to the point of beginning. On the next throw this is reversed and this process is kept up, commencing alternately at the end and beginning of dump track.

Contractors with a "dinky" equipment have a dump crew of 8 men and a boss to shovel off and throw track. With standard equipment the dump crew only shovel off and the track crew, consisting of 14 to 22 men, throw the track. In dry weather a 14-man crew can throw a 60-pound track. In wet weather and heavy work with sticky dirt, up to 22 men are needed. One property that had considerable difficulty in maintaining its dumps had 6 men and a boss on its dumps and 26 men on the track crew. The dump crew handled about 400 cars, a minimum of 2,000 yards, on a shift. Two dump crews, working 2 shifts, keep one track crew of 26 men busy one shift. The best record made on this property is 1,000 cars on 2 shifts, 33 men on dump and track crew.

Muskeg swamps abound in this section of the country. They may look inviting to the inexperienced eye, but make very unsatisfactory dump grounds. The strippings disappear and the muskeg bulges up. Most of these swamps seem bottomless. It is impossible to keep up tracks over them; they bulge, slide sideways, and crowd up in front. When they can be crossed at all, delays are so numerous and vexatious as to render them well nigh useless. Swamps may often be used to advantage during the very cold winter months and sometimes after a few seasons' use make fairly satisfactory dumping ground.

Bank and boulder blasting.—The regular overburden is a glacial drift often containing large boulders ranging from 2 to 12 feet in diameter. The boulders increase in size and frequency near the bottom of the overburden. The lower benches often consist of boulders and a compacted clay and other fine material, having the effect of a solid rock mass. This must be loosened up before the shovel can make any headway. As the shovel encounters these large boulders in the loosened bank, they are "chained out," moved to one side and a regular trail of boulders is often left in the wake of the shovel. They are then drilled and blasted by "single-jackers."

The benches are drilled and blasted by a regular crew of "gopher-holes" consisting of 10 to 30 men, common laborers, working in gangs of two. The benches, which range from 15 to 25 feet in height, are riddled with a series of holes 15 to 25 feet deep, spaced 15 to 25 feet apart. The collar of the hole is at the base of the bank and the hole points downward at an angle of 15° to 20° minimum spacing for high banks and hard material, maximum spacing for the lower and softer banks.

These holes have a diameter of 14 inches. Drilling through the loose stuff is accomplished by an ordinary round-pointed shovel blade with sides slightly turned up, provided with a long pole handle, 20 to 25 feet long, with 2½-inch diameter. As the ground becomes harder, 24-foot auger drills are used. If the material needs loosening up, a stick or two of dynamite is dropped in and exploded and the loose

material removed with a shovel. Drilling time for such a hole varies from 2 to 12 hours, according to the ground; wages, $2.00 per 10-hour day.

Du Pont black blasting powder is used, from 6 to 8 kegs (25 pounds) to a charge. Two men constitute a loading crew; they load from 3 to 7 holes in a day. The powder is loaded into the hole on a rectangular spoon made of 1-inch lumber, 32 inches long and 3 by 3 inches inside dimensions, fastened to a 25-foot handle. To avoid explosion of powder from possible sparks, copper rails are used in construction of spoon. The Oliver Iron Mining Company have recently abandoned this method and developed an apparatus consisting of a long wooden launder, 3 inches square, into

FIG. 57. Stripping Buffalo and Susquehanna Mine.

which the powder is fed through a hopper. A hand blower is attached to the launder by a rubber hose. As the powder is blown into the hole the launder is gradually pulled out. The detonator consists of 5 sticks of 60 per cent dynamite tightly wrapped together, two of which have electric caps. This bundle is rammed in tight with a long 1-inch pointed stick. The balance of the powder, usually 2 to 3 kegs, is rammed in tight; the hole is tamped with loose gravel to the mouth; tight tamping is essential. The holes are fired in batteries of 3 to 14, usually 5 to 8 at a time.

The large boulders remaining in the wake of stripping shovel are very hard

granite and the cheapest way to handle them is by drilling and blasting. This is done on contract at 25 cents per lineal foot. The holes run from 2 to 3½ feet. A good single-jacker will average 12 to 15 feet per day on contract work, making from $3 to $3.75. He will do more than a couple of men working double hand on company account. The blasting is done by company men. Two to four sticks of ¾-inch 60 per cent dynamite is used to the hole. The powder is well rammed and tamped with a little moist clay.

Taconite when broken up, though in one sense harder than regular banks, is shovelled without blasting since the teeth of the shovel can get hold and lift it up. Small patches of hard, solid taconite are often encountered. They are blasted by putting down a line of vertical holes in the bench ahead of the shovel, 8 to 10 feet from the face and 6 to 10 feet apart depending upon the ground. These holes put down with a jumper drill and heavily charged with high-grade dynamite.

Tracks, etc.—In open-pit mining, the approach to the pit and the general track layout must be carefully studied, the following requirements being kept in mind:

1. The approach through which the tracks enter the pit should be as convenient as possible both to the stripping dumps and to the main line. Sometimes a temporary stripping approach is cut for all or part of the stripping work. It is desirable that the stripping haul be as short as possible without exceeding desirable grades.

2. The approach should, with a maximum 2 per cent compensated grade, render the bulk of the ore-body (or that portion of it to be mined by direct open pit) readily accessible without necessitating unduly steep pit tracks. On large work, standard gauge track and 80-pound rail is generally used. The main pit tracks are carefully graded, lined in and ballasted, and properly maintained. No absolutely set rules are followed as to grades and curves. They are made as light and easy as conditions permit. As a rule 7° 30′ is considered the maximum desirable and 15° the maximum allowable curve. There are many instances where the exigencies of the case compel sharper curves. In fact 50° curves are occasionally unavoidable and do not give a great deal of trouble when properly laid and maintained.

On approaches a 2 per cent compensated grade is considered the maximum allowable, though here again circumstances sometimes compel a 3 per cent grade. In the pit grades are held down if possible to 2 per cent or at the most 3 per cent for main tracks. At times even 5 per cent is unavoidable for short runs, in which case a 19 by 26 locomotive requires a "pusher" to help it take 2 ore cars up the grade. Special "return" or "empty" tracks are sometimes laid on 5 to 6 per cent grades to permit the empty train to return quickly into the pit, the loaded trains going out by the more circuitous route over easier grades.

The stripping and dump tracks are often put in by the track boss in a hit or miss fashion without careful leveling or lining in. This is also true of the temporary loading tracks in the pit. It is very hard to get a steam shovel to cut a definite grade and their grades can only be approximately maintained. For ore tracks the ideal system is the ellipse or the spiral. These usually give light grades, easy curves and

turnouts; and the minimum of switchbacks. A pit must be large and uniform in order to apply this system. Long, narrow pits or pits with irregular ore-bodies could not be worked on this plan.

A mile of standard 80-pound track represents an investment of from $9,500 to $10,000 as follows:

TIES—	
Tamarack ties, lasting two years on stripping track which is thrown and about five years on stationary tracks, cost 45 to 50 cents, laid down. A mile of track requires, roughly, 17 ties per 30 feet of rail, or 3,000 ties per mile.	
3,000 ties at 50 cents	$1,500.00
RAILS—	
440 80-lb. rails, making 126 long tons (2,240 lbs.) at $35 per ton	4,410.00
6,000 tie plates (four spikes to the tie) at 8 cents each	480.00
880 splices (42 lbs. to the pair) at $2.50	450.00
1,760 bolts = 6 200-lb. kegs, at $3.50	42.00
Cost of supplies	$7,032.00
Cost of laying track, leveling and tamping, per mile	2,000.00
Total cost per mile, exclusive of making grade	$9,032.00

Drainage.—In open-pit mining the drainage question is often quite an item of expense. Many pits handle from ½ million to 1½ million gallons per day, requiring a pump shaft on the edge of the pit and sometimes a system of drifts beneath the pit.

THE STEAM SHOVEL

The last ten years have witnessed many improvements in the steam shovel. The service demanded of it on stripping work and direct mining from the face subjects it to continuous hard usage and the severest of strains. The modern steam shovel is really one of the mechanical wonders of the age.

Requirements.—The shovel must be able to stand up to the severest work 20 hours out of the 24, six days in the week, month after month, with the minimum of breakdowns or delays. The shovel is the mainspring of the job and the subject of breakage and delay at the shovel is the most serious that confronts the mine superintendent. Sunday is usually devoted to cleaning and repair work and once in two years the shovel goes to the shop for a general overhauling.

A good shovel is the result of intelligent design based upon thorough understanding of the service demanded that comes only from years of experience. Such a shovel is characterized by simplicity and strength. Every possible strain must be allowed for, with an ample safety factor. After considering all the strains that can be calculated there still remains the fact that the dipper may at any time and in any conceivable position strike with all its power a huge boulder or a badly blasted layer of taconite which it is powerless to dislodge. The resulting concussion is tre-

mendous and the strain can not be calculated. This very point has resulted in checking the tendency towards larger and more powerful shovels. The 95- or 100-ton shovel is to-day considered the standard for heavy stripping work. The advisability of using larger shovels is very questionable. The larger and more powerful the shovel, the greater chances will the runner take. Working against a high bank, he meets a hidden boulder and puts on force enough to either bring out the boulder or break down the shovel. In the hands of any but the most experienced and careful runners its greater strength would be a menace that might easily offset its greater capacity. The repair bills on the few large shovels in operation would tend to bear this out.

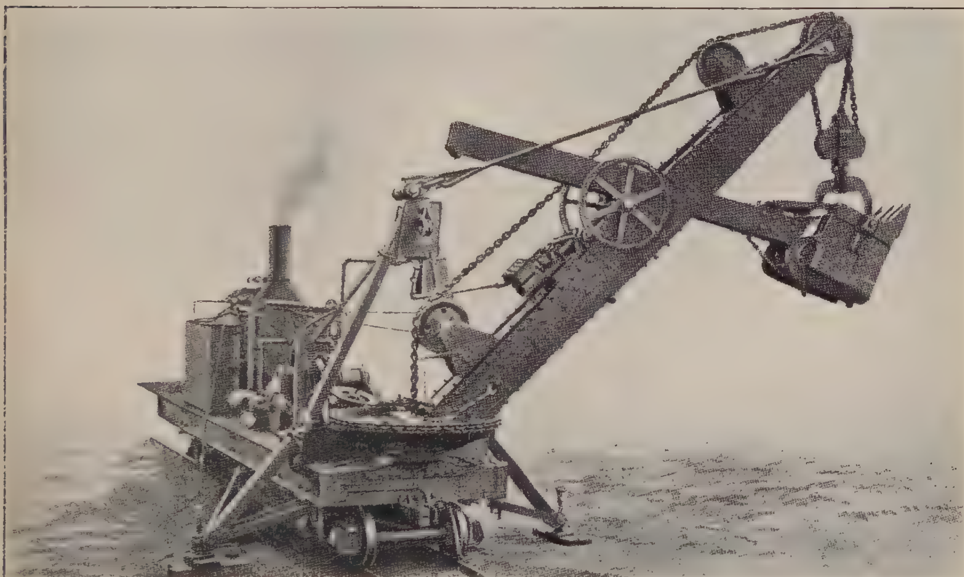


FIG. 58. Marion Shovel Stripped.

A shovel to be safe should be underpowered. The engine, though strong in itself, should be the weakest part of the machine. If all the other parts are proportioned safely to withstand any strains that may result from encountering a relatively immovable object under maximum steam pressure with throttle wide open, then breakdowns will be a minimum.

For stripping and mining work a shovel should be able to load standard ore cars standing on a track at least 6 feet higher than the shovel tracks. Then the work can be taken down in benches of not less than 12 feet and a through track maintained for the cars. A 12-foot bank is the least height a shovel can work in to advantage, and a 20- to 25-foot bank is much better. On anything less than a 12-foot cut the cross-section is so small that too much time is lost by frequent moves ahead; besides the dipper does not fill so well. To load into flat cars on a track 6 feet higher

than the shovel track, the dipper should have a clear height of at least 14 feet from the rail on which the steam shovel stands to the underside of the dipper door when open in its highest position. For economical operation this height should not be the extreme limit of the shovel, but there should be a foot or two to spare. Much time is lost if the runner has to crowd the dipper up to the last few inches to get his load on the car.

Types of shovels.—The shovel equipment of the Missabe consists largely of the Bucyrus and Marion makes. Both shovels are of the chain-hoist type as illustrated by Fig. 58, which represents a Marion shovel stripped of its housing.

The model 91 Marion, working weight 120 tons, equipped with a 3- or 5-yard dipper, and the 95-C Bucyrus, working weight 107 tons, equipped with a 3½- or 5-yard dipper according to requirements, are largely used on standard stripping jobs. The Bucyrus 95-C steam shovel has the following specifications:

Weight in working order, 107 tons. Clear lift from rail to bottom of dipper door when open, 17 feet. Extreme height from top of rail to point of boom, 28 feet 9 inches. Extreme height of A frame, 19 feet 4 inches; can be lowered to 14 feet 6½ inches. Width of cut at 8-foot elevation, 66 feet. Dimensions of car over all, 44 feet 2 inches by 10 feet. The cost of this shovel varies from $12,500 to $13,000 f. o. b. Milwaukee according to the price of raw material. The car body is shipped on its own wheels and the boom, dipper, dipper handle, jack arms, coal platform, propelling sprockets and chains, etc., are all removed and loaded into a flat car. The cost of shipment to Missabe points is, roughly, $450.

For lighter stripping work and stock-pile loading, the 70-C Bucyrus is quite largely used. The same machine is in general use on Panama Canal work. Its working weight is 83 tons; clear lift, 16½ feet; width of cut, 60 feet; over-all dimensions, 36 feet 4½ inches by 10 feet. Cost, $9,000 to $9,500, with $325 approximate freight charges.

Within recent years the Atlantic Equipment Company, of New York, have introduced a radical departure in steam shovel design which has met with great favor on railroad and general contracting work. It is known as the Robinson shovel. The principal departure consists in hoisting by direct wire rope with one sheave instead of by chain and multiple sheaves as is usual. Direct wire rope hoisting has for several years been a success in large dipper dredges, but its application to steam shovels seemed impossible because of the difficulties encountered in passing the wire rope over so many sheaves. This is overcome in the Robinson design by mounting the hoisting machinery directly upon the boom, eliminating the many guide sheaves, and applying the power direct. There is also a change in dipper construction. The bail and sheave are discarded and the double hoisting rope is attached directly to the back of the dipper (see Figs. 59 and 60). Among the advantages gained by this construction is a decided increase of net lift for same length of boom. The Atlantic shovel of 110-ton size presents a very different appearance from the type of shovel commonly seen on the Missabe. The Oliver Iron Mining Company have a few of these machines in use. That they are not in general use may be attributed to the

FIG. 59. General View of Atlantic Shovel.

FIG. 60. Stripping with Atlantic Shovel.

fact that they are a recent innovation and that most of the mining companies have a full equipment of the other types. The manufacturers of the Robinson shovel present the following comparison between chain and wire rope hoisting types. The expressed opinions of a number of users of these shovels seem to a large extent to bear out these claims.

The measure of efficiency of a shovel is the digging power produced at the teeth of the bucket relative to the theoretical pull due to the pressure of the steam in the engine cylinders. The two principal factors which determine this are: First, the loss by friction in the mechanism of transmission; and, second, the angle of lead when the dipper is in effective working position. The internal friction of the engine in the Robinson is the same as in any shovel, but a great reduction in loss by friction is effected by exerting a direct pull with wire rope around a single sheave of large diameter as compared with an indirect pull of chain around a small drum and from 5 to 7 sheaves of small diameter. This friction amounts to about 40 per cent in the chain hoist and 10 per cent in the wire rope hoist. In a shovel working at four-dipper loads per minute and a hoisting chain passing over six sheaves, each link of

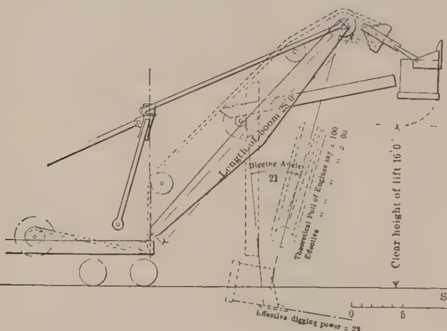


FIG. 61

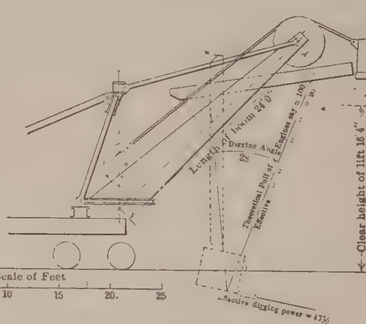


FIG. 62

the chain makes two bends every time it passes a sheave, once going on and once coming off. This is equal to 12 bends per lift hoisting, and 12 bends in lowering, or 96 bends per minute. If there be, say, 300 links in the chain passing these sheaves, this is equal to nearly 30,000 bendings (or slidings) per minute of one link on another, without counting the bends going on the drum. This accounts for the great friction and wear that takes place in the hoisting chain. The wire rope, on the other hand, with single sheave, makes two bends hoisting and two bends lowering four times per minute, equal to 16 bends per minute. These bends occur over a sheave of very large size which do not overstrain the wires of the rope or cause appreciable wear. The effective pull upon the dipper is therefore practically the same as the effective pull at the engines after internal friction of the engines is deducted.

The second important factor is the digging angle. In Fig. 61 the digging angle of the ordinary construction with chain hoist is shown, and in Fig. 62 the digging angle of the wire rope shovel with wire rope hoist is shown; from which it will be seen that a great increase in digging power at the teeth of the bucket is obtained

with the same initial pull in each case. This means that strains upon the dipper handle and boom are much reduced and the thrusting power required is less, while the shovel performs hard work with greater ease and less effort. Added to this is the great advantage of increased height of lift which is obtained with a boom of shorter length. The boom of the wire rope shovel is from 4 feet to 6 feet shorter for the same lift than in the chain construction, or, on the other hand, a proportionately higher lift can be obtained with the same length of boom.

The improved efficiency of the wire rope and the improved digging angle being taken together, the result shows that this shovel is from 65 per cent to 85 per cent more efficient than the best modern chain hoist shovels.

If the theoretical pull of the engines is assumed to be 100 in each case and we deduct 10 per cent loss by transmission in the wire rope shovel and 40 per cent for chain friction in the chain shovel, the pull on the bucket will be respectively 90 and 60 per cent. Resolving these in each case gives digging thrust and dipper-arm thrusts as follows (assuming vertical position of dipper handle in each case):

	Pull of engines	Pull on dipper	Dipper-arm thrust	Digging power
Chain shovel	100	60	57	22
Wire rope shovel...............	100	90	81	43½

This means that on the above basis the Robinson shovel exerts 40 per cent of the theoretical engine power at the teeth of the bucket, while a chain shovel with the above angle exerts 22½ per cent. It is therefore nearly twice as efficient.

STRIPPING WITH THE STEAM SHOVEL

Operating conditions.—A great many varying factors enter into the consideration of steam shovel operating conditions. A favorable combination of these factors will make for surprising efficiency, whereas an unfavorable combination will result in an equally surprising drop in efficiency. Under good operating conditions the steam shovel will load at from 1/20 to 1/30 the cost of hand work, while under adverse conditions it may cost as much, if not more, than it would to load by hand.

The proper function of the steam shovel is to cast up or load into cars loose or broken material. In stripping on the iron ranges the shovel is usually required to dig compacted and imperfectly blasted material, a much more strenuous service than mere scooping and lifting. The extreme irregularity of the service demanded makes it difficult to make serviceable average statements regarding performance and cost.

In stripping work the main thing is to keep the shovels (the source of supply) going steadily. There are inevitable interruptions in operation of sufficient frequency to make serious inroad on the gross available 10 hours. These delays are quite apart from the special interruptions due to breakdowns at the shovel or enforced waits due to trouble or delay in one or more of the auxiliary operations.

Stripping.—The essential requirements of a successful stripping job are:

1. Proper supervision of the whole job, with this thought borne in mind, that coöperation all along the line is the key to success.

2. First-class machinery adapted to the work.

3. Facility for quick repairs and thorough up-keep of the plant.

4. Skillful runners and cranemen, experienced in the particular field, preferably men accustomed to work together.

5. Easy access to shovel for stripping trains, the empty train pulling in as loaded train pulls out. This is accomplished by providing a "run-around."

With standard equipment and a single 23-foot width of cut this is not always possible, for if the stripping bank caves at all there will be no room for the double track. Similarly, on the top banks the empty trains must wait in the switch until the loaded train passes by. The shovel cleans up the corner (i. e., dirt spilled), while empty pulls in.

6. Good track systems, reasonable grades and curves.

7. A good pit crew and a well-organized dump gang. Delays and accidents on the dump mean delays all along the line and result in hanging up the shovel.

Maximum efficiency at the steam shovel can only be maintained when every other operation is considered subservient to the steam shovel which is the source of supply. Thus, on a properly organized stripping job half a dozen trains may be waiting on the steam shovel in various places and positions; the shovel should never have to wait on a locomotive.

Bank blasting is a very important part of the work and should be kept well ahead of the shovel. The measure of the efficiency of the blast is not the minimum powder consumption, but thorough comminution of the bank, giving maximum digging and loading efficiency at the shovel with a minimum of repairs. Poorly blasted material increases the strain on the shovel, the number of delays and breakdowns, and the repair bill, and causes both a direct and indirect decrease in output. It often pays to have emergency men in the pit whose labor considered as a unit may appear very unprofitable, yet they may serve to increase the net working time of a shovel and therefore of the whole organization, to the point of materially decreasing the cost per yard. This is especially true on rough work in rock and boulders.

Missabe stripping work affords all kinds of digging from sand to hard taconite. On the one hand is found straight sand having very little if any cementing material and small boulders. On the other hand, a sticky, clayey material that lodges between the teeth so that they can hardly penetrate the bank, clogs the bottom of the bucket and necessitates frequent stops to clean the bucket. On some jobs 10- to 20-foot layers of clay containing huge boulders up to 6 feet in diameter make some of the hardest digging imaginable. In such ground the teeth do not readily grip the smooth round boulders and slip in the clay; there is much violent tugging and severe strain on the machinery; the teeth last from 1 to 6 days and there are frequent minor breakdowns that take out 10 to 30 minutes for repairs. The large boulders are "chained out" and left in the wake of the shovel, afterwards to be blasted to convenient size. Taconite, unless thoroughly blasted, makes very hard and slow digging. In stripping estimates it is customary to estimate 1 foot of taconite as equivalent to 3 feet of ordinary overburden. On the more difficult work the pit crew

is larger, the net operating time per shift is less. More time is required for a complete swing of the dipper and the net load is usually less. Finally, the maintenance or repair bill is likely to be 2 to 3 times as much.

The Model 91 Marion or the 95-C Bucyrus is generally used for both stripping and mining; on stripping work their capacity ranges from 45,000 to 80,000 yards per month, working 52 shifts of 10 hours each. Seventy-five thousand (75,000) yards is an excellent season's record (one shovel, 52 shifts, 26 days per month), while 60,000 would be a good annual record including the winter months. The stripping record is in the neighborhood of 4,600 yards for one shovel, one day (2 shifts). The following reports show what is being done on the Missabe with standard equipment:

90-Ton shovel	Locomotives 19 by 24	Yardage per month, 52 shifts	Yardage per shovel per month
2	6	160,800	80,400
3	10	256,060	85,300
3	8	238,000	79,300
3	10	231,500	77,200

The life of a steam shovel depends largely upon the treatment it gets and the repairs put on it. A shovel working steadily on stripping in the pit for two years with necessary daily and special Sunday repairs is ready for a general overhauling in the shops. It is good policy to keep a shovel in shape to do maximum digging work all the time. The average annual repair bill when on stripping work ranges from $3,500 to $6,000, depending upon the rock and upon the skill of the runner. With proper care a shovel will last 12 to 15 years.

The partial records of one large standard stripping job show that on winter stripping with about 60,000 yards to the shovel per month, the repair cost is about 10 cents per yard moved, roughly $1.10 per actual operating hour, and nearly evenly divided between labor and materials. Many companies are equipped to manufacture most of the wearing parts of a steam shovel, in fact they can make everything except the boilers and engines and certain patented parts. The cost of some of these parts may be of interest.

Steam Shovel Trucks—

Labor	$ 96.00
Materials	400.00
Total cost	$496.00

Hoisting Gear, Complete—

Labor	$ 85.00
Materials	420.00
Total cost	$505.00

Two and one-half-yard Dipper—

Labor	$100.00
Materials	240.00
Total cost	$340.00

Dipper Handle—

Labor	$ 56.00
Materials	282.00
Total cost	$338.00

Dipper teeth.—Dipper teeth are repaired as they wear, that is they are repointed. They break very seldom, but wear down to a point when they must be discarded.

Cost of manufacturing 13 teeth:

Blacksmithing	$78.00	
Machinist (drilling)	6.00	
		$84.00
Materials		60.00
Total		$144.00

Cost per tooth, $9.60

Operating details.—There is of course much variation in the detail of operation at the face. In very hard clay and boulder digging or in hard, poorly blasted taconite the performance of a shovel may drop down to 750 or 1,000 yards in 10 hours, depending upon the height of cut. In average digging 1,500 yards, and in specially easy digging 2,000 yards may be considered good average daily performances.

The shovel crew will range from 8 to 10 men—

1 runner; wages, $5.77 per 10-hour day

1 craneman; wages, $4.04 per 10-hour day

1 fireman; wages, $2.50 per 10-hour day

4 to 7 pitmen (usually 4); wages, $2.35 per 10-hour day

In addition to the above wages, the following bonus schedule is adopted, based on continuous good service:

Runners, $25 per month of 26 days

Craners, $20 per month of 25 days

Locomotive engineers are paid $4.10 per 10-hour day with a $20 maximum bonus.

The direct labor cost at the shovel therefore varies from $23.45 to $30.50 per day, assuming that the full bonus is paid. The time consumed in the various motions of the shovel vary with the nature of the material and the height of bank. On a low bank the dipper does not fill easily and two or even three lifts are necessary. On a 30-foot bank of good digging material, the dipper will load and complete its swing in 20 seconds. In difficult digging or low banks double the time may be consumed. The average time ranges from 25 to 30 seconds for a complete swing.

The time required to load a train is subject to greater variation. It varies first of all with the average load, and the time of swing per dipper, and, secondly, because the shovel may have to "move up" during loading time of a train. It may take anywhere from 10 to 25 minutes to load a train. The distance moved up in stripping is usually 5 to 6 feet, and it takes from 3 to 5 minutes for a shovel to move up. On a 30-foot bank a shovel will move up about 10 times a shift, once an hour. This gives the

pitmen ample time to clean up, lay track ahead of the shovel, and get everything ready for the move. Under these conditions the minimum time is consumed in moving up. On a 20-foot bank a shovel will move oftener, perhaps on an average 15 times a day. On low banks and on "clean-up" work from 20 to 30 moves are necessary, thereby consuming much time.

The examination of a number of records shows so great a variation in various operating details that no average statement can well be made. The actual loading time varies from 40 per cent to 75 per cent of the 10 hours available. The principal delays are: "Moving up" of shovel, which takes out from 25 per cent to 5 per cent. Repairs on shovel, which take out from 15 per cent to 10 per cent. Waiting on cars, i. e., delays in transportation system, 15 per cent to 5 per cent. This still leaves considerable time unaccounted for to be classed as miscellaneous delays. The principal causes of delay outside of "moving up" and changing of trains are: Blasting; difficulty in filling dipper due to hard bank, boulders, or low bank; chaining boulders; shovel off track; shovel truck slipping; chain, pin, or teeth breaking; clogging up of dipper; poor coaling arrangements.

On a well-organized stripping job, with standard equipment, eight 12-car trains serve 3 shovels that average 5 hours' actual loading time and fill on an average 250 cars each, giving 1,500 yards' average duty per shovel per 10-hour shift. The net time required to load a 7-yard car with 6-yard actual capacity, is 1.2 minutes. The yardage per man on stripping work, i. e., including every man on the job, varies from 10 to 20 yards per man on day shift, and 20 to 30 yards per man on night shift. All the incidental and repair work is done in the day time; there are no interruptions on the night shift except from breakdowns; hence the yardage is from 30 per cent to 50 per cent greater.

A stripping shovel will use the following supplies on a normal job:

Coal per shift of 10 hours.................................2½ to 3½ tons
Lubricants—
    Black oil ....5 gallons, 24 hours
    Cylinder oil ....5 gallons, 24 hours
Illuminants—
    Gasoline ....10 to 15 gallons a night
    Kerosene ....2½ gallons a night
    Water ....12,000 to 15,000 gallons per 24 hours

ORE MINING WITH STEAM SHOVEL

On ore the variation in duty of a shovel is even greater than on stripping work. The physical condition of the ore, the condition of the shovel, the skill of the runner and the care he gives his shovel, weather and track conditions, regularity of ore-car supply, all have a decided bearing on the individual performance.

Aside from mere loading the steam shovel sorts out bunches of ore, makes grades, mixes ores to get uniform silica and other contents, and so forth. The limited area of some pits and the consequent grades and curves make it a difficult matter to so

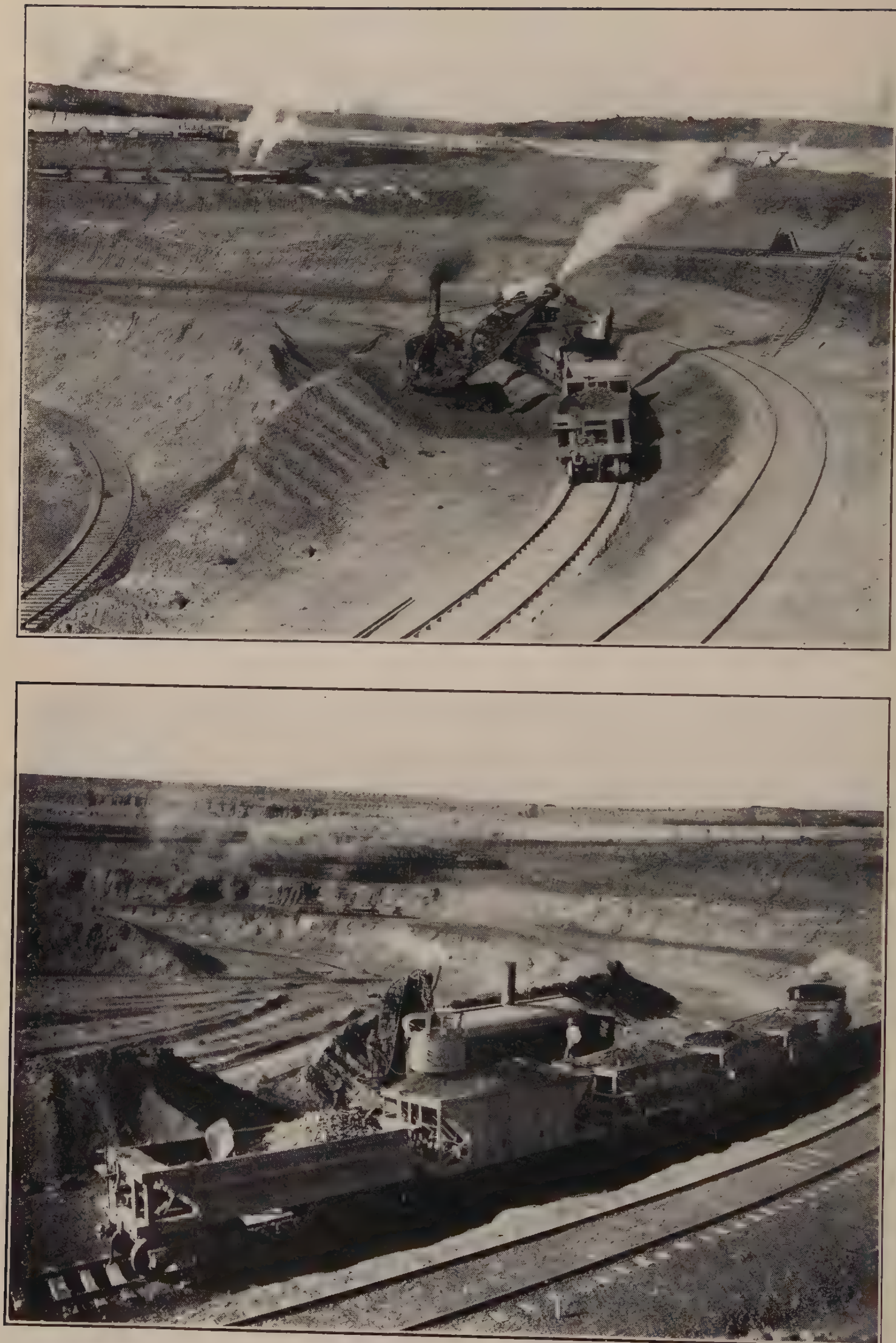


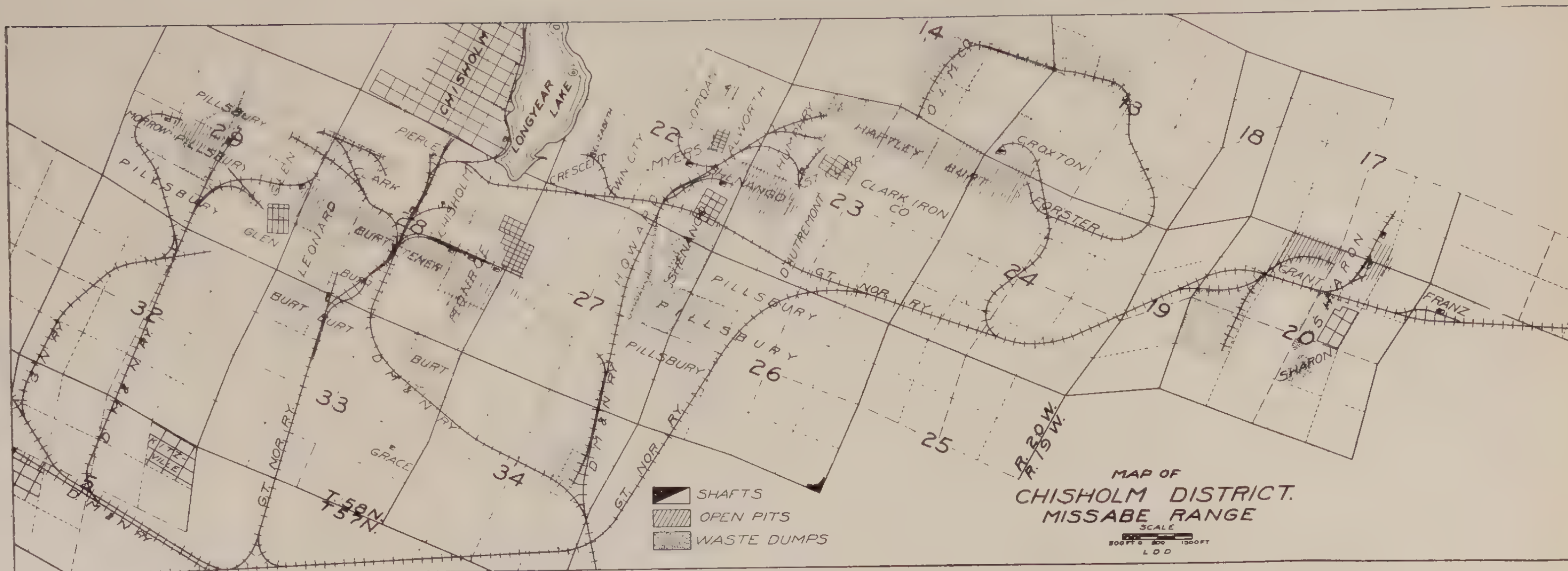
FIG. 63. Mining Ore with Steam Shovel.

arrange the trackage that the shovel is always kept busy. Shovels often work under such adverse conditions as 4 per cent and 5 per cent grades from loading track to main track, 30° to 50° curves, narrow pits and difficult approaches, all of which greatly retard the ore-trains and lessen the out-put of the shovel. As a general statement it may be said that 75,000 to 100,000 tons is an average month's work (52 shifts) for an ore shovel throughout the shipping season. This allows for ordinary delays. One hears many statements about keeping shovels running continuously. As a matter of fact there is quite a proportion of idle time due to breakdowns, wrecks, changing location, making desired grades, car shortage, and other causes. It may be said that 60 per cent would be considered well above the average running time. When there is a great deal of sorting to do, the tonnage may be much smaller. On the other hand, shovels often run 100,000 to 125,000 tons a month. Daily performances of 4,500 to 6,000 tons on 2 shifts are not uncommon, but can not be maintained over a season. The record for a day's work (2 shifts) is, I believe, 11,000 tons. The record for a month's work is 130,000 tons. Fig. 63 is a general view of a steam shovel loading ore direct from the banks.

The shovel crew consists of runner, craneman, fireman, 4 pitmen, and a varying number of "rock-men," the latter ranging from 2 to 10. If the ore breaks out in large chunks, it must be sledged. If taconite occurs, it must be picked out; 4 to 6 rock-men are often used in the pit and a couple on the ore-cars to throw out rock or poor stuff. The shovel throws back whatever rock the pitmen can not handle. These chunks are thrown down near the loading track at the foot of the bank for later loading on dump cars. Several dump cars are left at the shovel. When an ore train comes in to be loaded, they are coupled on and pushed along with the train. When the train is loaded, the dump cars are uncoupled, i. e., "spotted" at the shovel. The latter then picks up the rock and loads it. Any rock the dipper can not readily pick up is thrown into it by hand. The next ore train pushes the cars away; this performance is repeated. Sometimes rock is loaded into ore cars with the ore. Two men pick it out as fast as possible, sometimes delaying the dipper to some extent.

Considerable ore is spilled during the loading and a small bank forms in the pit beside the loading track. When there are no rock dump cars ahead, the shovel "casts ahead," i. e., picks up this ore and throws it in front of the ore face, as soon as the loaded trains move out. In case the space beside the shovel is occupied by rock cars, there is a loss of time while the ore-train bumps the rock cars out of the way, gives the dipper time to clean up, and then returns to be loaded. The ore bank in some cases is soft enough to be dug by dipper teeth without blasting. In most ore-bodies, however, blasting is necessary. This is done by putting down jumper-drill holes 8 to 10 feet apart and 15 to 20 feet from the edge of the bank. Their depth varies from 12 feet to 24 feet. The holes are chambered or sprung with a small charge of dynamite and then loaded with 2 to 3 kegs of black powder (25 pounds to the keg). From 3 to 5 holes are shot at a time, usually at quitting time, though when it is necessary they are fired at any time during the shift. The result of the blast is merely a shaking up of the bank; the face is rarely thrown down. After

FIG. 64. Two Views of Adams Pit in 1909.



District opened in 1897.
Number of Shipping Mines 19.

Approximate tonnage shipped 26,000,000
Approximate reserve tonnage 225,000,000

FIG. 65. Map of Chisholm District.

a blast the dipper is used to "claw-down" the face. The dipper with its bottom open is dragged across the face and the material drops down in front of the shovel.

The actual working time of an ore-shovel is subject to extreme variation. One of the most frequent sources of delay outside of happenings at the shovel is congestion of ore-cars resulting in lack of regular supply of empties. The actual loading time of a shovel will vary from 2 to 6 hours. Modern steel cars of 100,000-pound capacity are used almost entirely, from 2 to 4 cars to an engine on steep grades, and 6 to 8 cars on easy grades. One typical property reports 3 ore-shovels, scheduled to fill on an average 300 steel ore cars in 24 hours, that is, 50 cars to the shovel per

FIG. 73. Milling System, Fayal Mine, early stages. Each little crater is the rim of a mill.

10-hour shift. These cars have a rated capacity of 50 tons. On account of the lightness of the ore at this property, a carload averages about 40 tons.

Fig. 65 is a sketch map of the Chisholm district, on which the locations of the open pits and their respective stripping dumps are shown to scale. Plates VI and VII contain a number of illustrations of open-pit mines, Figs. 66 to 72 inclusive.

THE MILLING SYSTEM

The "Milling System" is a combination of stripping and underground mining that may be advantageous under certain conditions. A hoisting shaft is sunk well outside

of the ore-body, and if there are to be two or more levels, the first level is established 50 to 75 feet below the top of the ore. The level is opened with a number of branch drifts either on the rectangular or on the diagonal plan. A portion of the ore-body is stripped and raises are run from the drifts to the top of the ore (now the surface). These raises are quite large, 40 to 50 feet apart and provided with a strong chute. The ore is simply mined ("milled") into these raises, loaded into cars on the level, and trammed to the hoisting shaft. When the process is well under way, the pit has

FIG. 74. Milling System, Fayal Mine, later stage. Small craters merged into two large ones.

the appearance shown in Fig. 73, which represents the early stages of the milling system in the Fayal pit. Each little crater represents a raise. As these raises are mined out a second series of raises is put up midway between the first to tap the pyramids of ore remaining between the raises of the first set. Fig. 74 shows a more advanced stage of the work. Fig. 75 shows the Albany Mine in 1909.

The miners work on the sides of the craters and securely fastened long ropes are kept within their reach to prevent them from falling into the raises in case of ore-slides. The method of breaking is to drill holes 5 to 20 feet deep around the

FIG. 75. Milling System Albany Mine.

FIG. 76. Enlarged view of right hand portion of Adams pit, showing steam s
loading ore into "mills."

Fig. 66—Mountain Iron Mine

Fig. 67—Mountain Iron Mine

Fig. 68—Mahoning Mine, Hibbing, 1910

Fig. 69—Stripping Leonard Mine, Chisholm, 1[illegible]
Underground ore being stock-piled in pit for subseq[illegible]

FIG. 70—Adams Pit (1909)
Showing old underground working in the bank and craters left by Milling System of mining

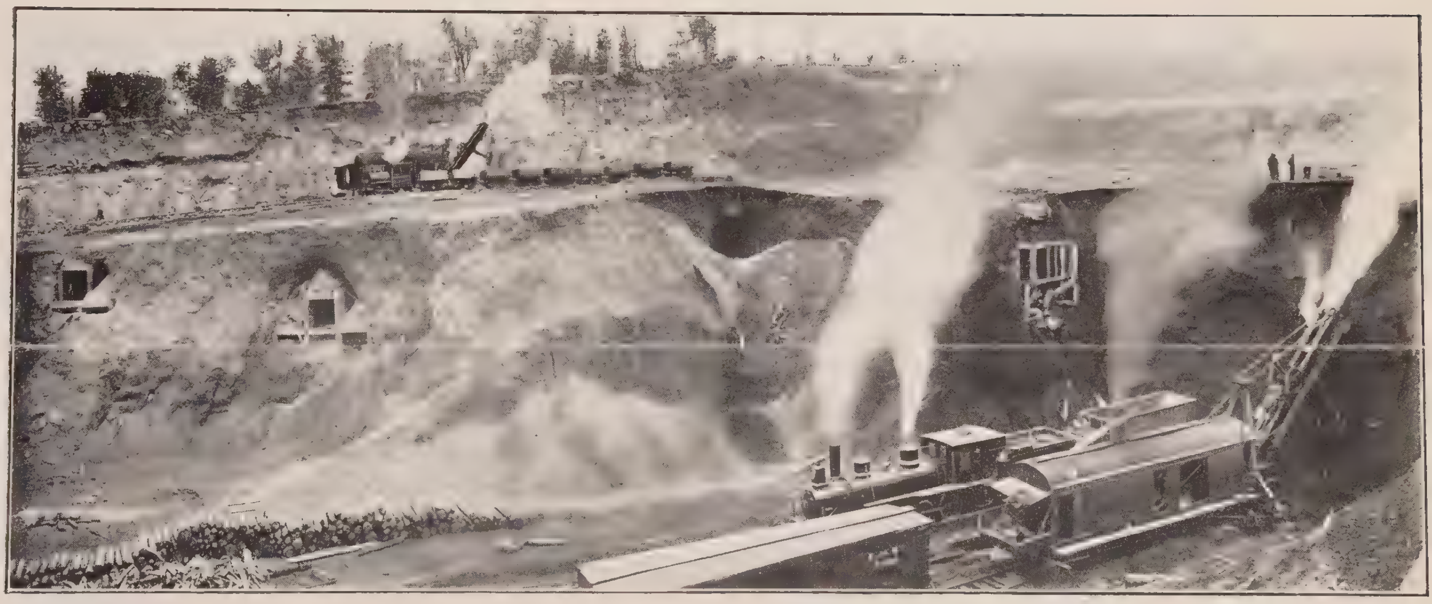


FIG. 71—Changing over from Underground to Open Pit Mining

FIG. 72—Shenango Mine in 1908

ides of the craters with pointed 1½-inch steel rods. These rods are provided with lamps near the top. They are driven with sledges; when they strike, they are arted by placing railroad jacks under the clamps. The holes are first sprung or hambered with 3 to 4 sticks of dynamite and then charged with from 100 to 300 ounds of black powder. The raises are often "hung up," or clogged, and this can nly be prevented by keeping large chunks from getting into them, not letting them ecome too full and drawing them down frequently so that the ore does not stand in hem too long.

The advantages of the milling system are manifest. A much smaller initial xpenditure is required in that the entire ore-body need not be stripped. A comparavely small amount of stripping will expose enough ore to begin production. When block of ore has been completely mined, the pit may be utilized as a dumping round for future strippings. Therefore, a much smaller waste dump area will ffice. The approach, often a heavy expense, is eliminated. Against these adntages may be mentioned the expense of a shaft and its equipment, and the cost the raises. On the other hand, an ore-body suitable to the milling system would doubtedly, if mined by open-pit method, require a drainage shaft. A further disvantage—and this is probably the greatest weakness of the system—is the danger flooding the mills with sand and slime during heavy storms. This can be to some tent controlled by extensive surface ditching, but is always more or less of a enace.

The ideal condition for the application of the milling system is a medium size e-body that would pay to strip, but would not admit of economical locomotive ulage of the ore. Under such conditions the cost of operation would be about dway between the costs that would obtain under open-pit and underground mining spectively. It is said that milling ore will cost from 10 to 20 cents more than en-pit ore from pits of reasonable capacity and 20 to 30 cents less than slicing ore der normal conditions. The tonnage per man per shift is quite large; it may run om 25 to 30 tons and more under especially favorable conditions. Most of the e is broken directly into the raises and practically no ore is lifted by hand until the ttom layer is reached.

In some deposits the steam shovel is used on the bottom layer to mine and dump e ore into the mills (see Fig. 76). In other cases a shovel is used to load into cars ich are then trammed to the hoisting shaft. This latter method is not milling, of urse, though it is sometimes called "steam shovel milling." Fresh shovel cuts are rted in pits of this kind by bringing in a drift from the shaft at the desired elevation d making an opening for the shovel to begin a new cut.

DEVELOPMENT OF AN OPEN PIT

In order to follow the development of an open-pit mine, a property was selected ich illustrates the various stages of the work. One-half of the mine is stripped l commencing to ship ore while two years' stripping remains to be done on the er half. Furthermore, the property offers a good example of what might be

considered ideal conditions which for business and other reasons could not be taken advantage of, particularly with reference to the disposal of the stripping.

The Minnesota-Wisconsin Mine comprises four "forties" running east and west. Operations were started in the spring of 1906 on the Minnesota eighty, the fee to which is in the operators. Subsequently operations were extended to include the adjoining Wisconsin eighty which was taken on a lease. This acquisition materially changed the plan of attack, inasmuch as in general it is the policy of most operators to mine out the leases first and leave fee ore intact. The maps shown on Plate VIII were traced from the operating companies' maps for the years 1906-10. Some additions and sketches were made to illustrate various phases of the problem.

Fig. 77 is a topographical map of the property showing 5-foot contours and the location of all drill holes. The drilling was done many years ago and the irregularity of spacing of drill holes is quite marked when compared with careful and exact engineering methods of to-day. Some later confirmatory drilling which adds to the irregularity was also done.

A glance at the contour map shows that the country pitches to the southward and eastward. This offers an ideal approach for the stripping shovels and trains on the east and an ideal dump on the south. The territory to the south, however, was not obtainable for this purpose. The available dump room at that time was limited to a comparatively small acreage on the northern portion of the Minnesota eighty. Preparations for stripping were made by grading in a track from D. M. & N. Ry. terminus (shown by heavy single line) to I. At this point the steam shovel was started making a "thorough cut"—a cut 23 feet wide and 7 to 8 feet face—"casting up" the dirt on both sides. This is the cheapest way of grading a roadbed on rough ground. On this roadbed the first track for stripping train was laid to what was then the western boundary of the property.

At the point II the first dump track was turned off. Some 900 feet of this was carried on a light trestle, 10 to 15 feet high, filled in with stripping to point III. The dump was extended by swinging down hill; its limits (see dotted outline) were soon reached, though it was subsequently extended from another track system and is shown on Plate VIII as stripping dump No. 2. To obtain more dump room a second switch was put in at IV and a switchback run on a 1.7 per cent grade from point IV to point VI, and on a 1.5 per cent from VI onward to make the first important dump. The grade was made comparatively light on account of a 15° curve which is sharper than desirable. This gave two short dumps that could be swung both ways, the completed dump being No. 1 on Plate VIII. Another switchback was put in later starting at point V and running on a 2.2 per cent grade to the natural surface where dump No. 2 was started. Later on the two dumps marked No. 3 were swung off. The steam shovel was started in May, 1906, the first cut being taken along the south stripping limit, subsequent cuts moving north. The yardage per month from May 1 to January 1, 1907, is indicated on Fig. 78. The total for this period is roughly 650,000 yards.

The stripping tracks during the first year were all down grade in favor of the

Piling brush preparatory to stripping. FIG. 84 First shovel cut.

View of pit during second stripping season.

Buildings. FIG. 85 Changing boom with wrecking crane.

load. The second year three shovels were at work and the last track had a maximum 2 per cent grade against the load, no switchbacks being necessary. The yardage for the year 1907 was nearly 2¼ million yards, the total yardage being a little more than 2¾ million. A permanent approach was also cut, 1,800 feet long, maximum depth 35 feet, and total yardage 85,000, an unusually small amount. The tracks in the approach are level; this is also unusual, since approaches generally have to be cut down decidedly below an economical grade. The shipping tonnage opened up by this stripping may be roughly stated as between 2¼ and 2½ tons of ore per yard of stripping.

The dumps already mentioned were utilized to their full capcity and the available space was quickly covered. The purchase of two "forties" north of dump No. 3 enabled the building of a 2,500-foot trestle, 20 feet in height, trestle No. 1, from which two dumps were started, one about 1,200 feet long, thrown or fanned out to the west, and north of it another dump, thrown both east and west. Later this same dumping ground was utilized for stripping from the adjoining Wisconsin eighty by bringing in a trestle on a 1.5 per cent maximum grade, marked trestle No. 2. From the junction of this trestle with the original dump grade the North dump, 4B, was filled out to dotted boundary, 4D. A new dump was thrown off eastward as shown by dotted outline and the original dump was also filled in from an extension of trestle No. 2 shown as track No. 2 on the northerly forty.

The dumps were becoming too short and the curves too sharp to throw track satisfactorily; so the intervening space between the north and southeast dumps was bridged by trestle No. 3 and filled in, uniting the two into one long dump which was extended eastward as No. 4E, as shown by dot and dash outline. The jog in the outline is accounted for by the fact that the forty adjoining the northern forty in use was not available for dump purposes. Meanwhile another trestle, No. 4, was built on top of ground filled in from No. 1 with a grade steep enough to give a 20-foot dump, shown in dot and dash outline 4F. The ground to the north is a swamp, available for a few months in the winter time.

The first year's stripping from the Wisconsin more than exhausted the available dump room and the disposal of the remaining 6,000,000 yards offered quite a problem. A plan was outlined to take trains out through the west end of Wisconsin over track system shown on Fig. 83 to point A, a distance of two miles and then by switchbacks utilize the space shown by dotted dump outlines covering four forties. This plan was abandoned for various reasons, among them the fact that trains must come out of the pit ahead of the engines. In dumping, trains must have the end car of train at end of dump so as to keep up the ends on the dumps.

Finally, property was acquired for the location of dumps 5 and 6. The 10-acre strip east of dump No. 5 was also acquired. A part of this ground was muskeg and presented many difficulties. There is a sudden drop of 55 feet in a short distance, requiring a 1,000-foot trestle on a 2¼ per cent down grade with an elevation of trestle of 25 to 35 feet. When this was filled in, a second trestle was built over the first and thus a solid track was finally built to the top of, and across, the end of the com-

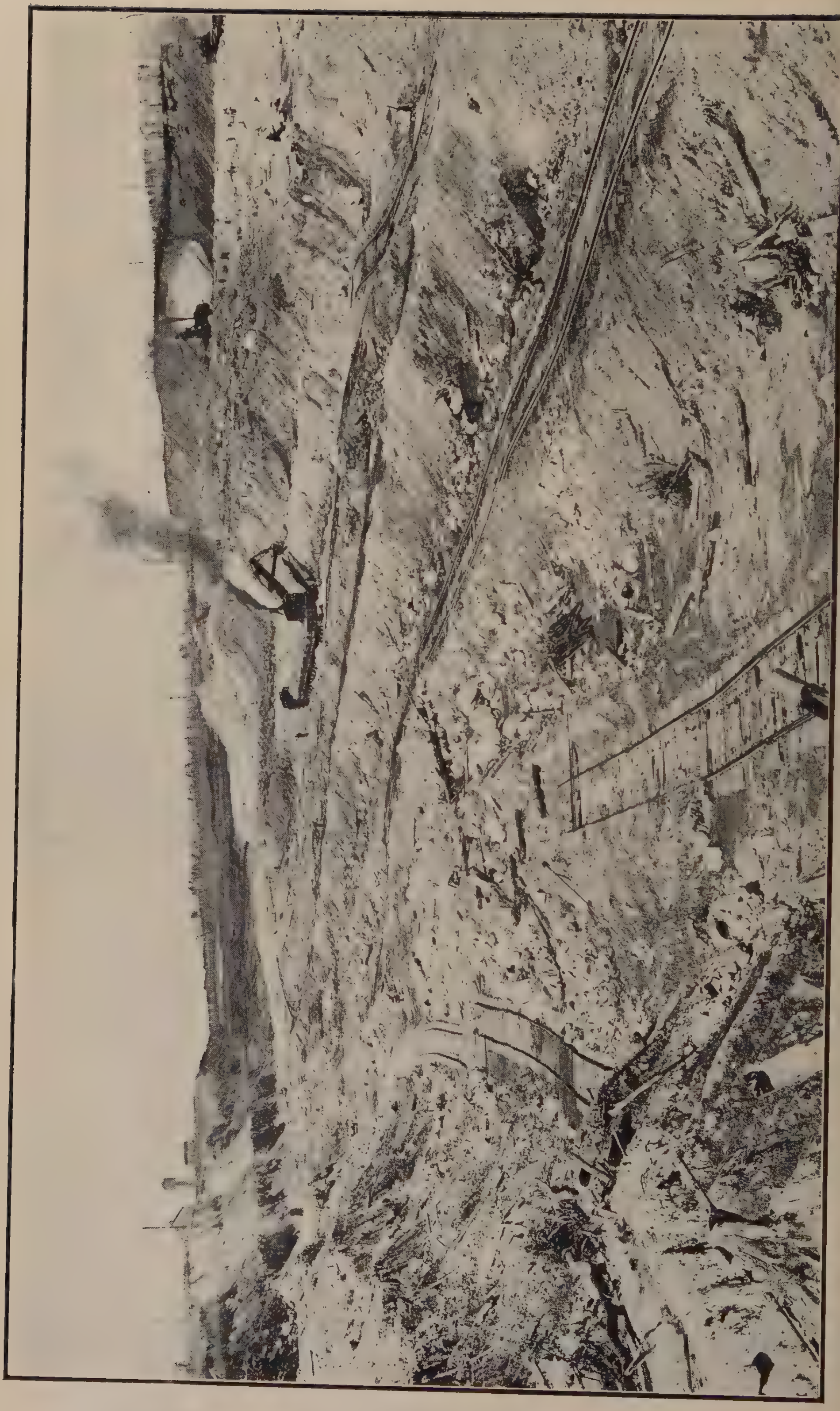


FIG. 77—Contour and Exploration Map

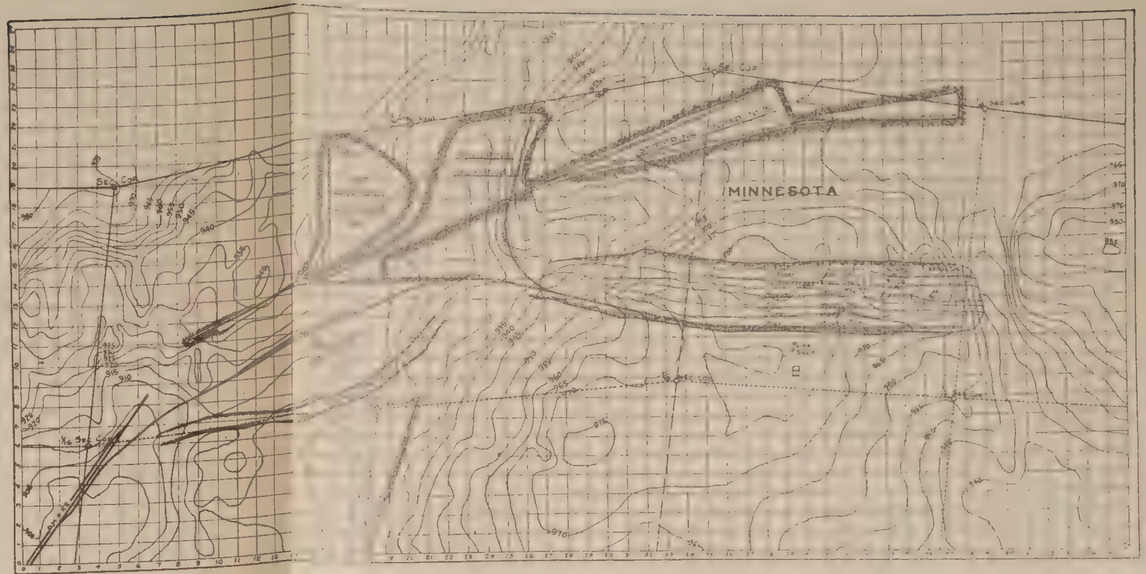


FIG. 78—First Season's Work

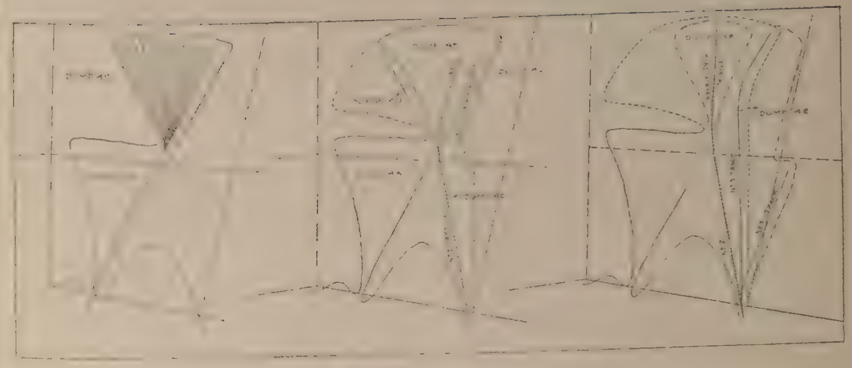


FIG. 79—Dumps

FIG 80—Second Year's Work

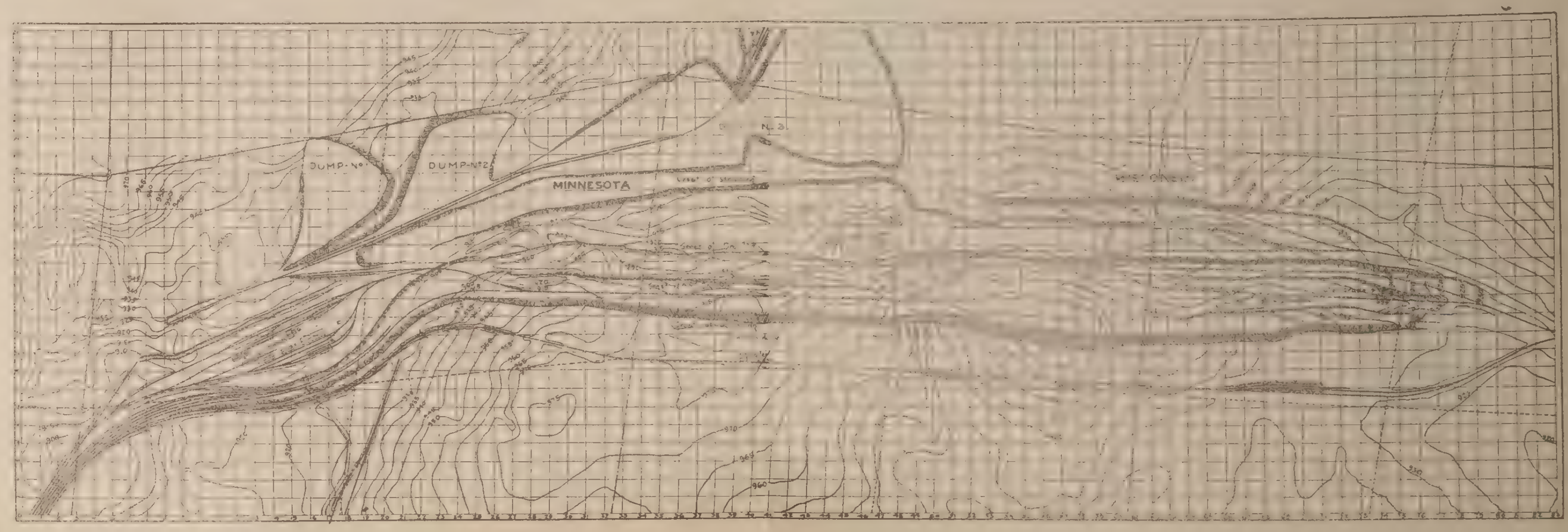


FIG. 81—Third Year's Work

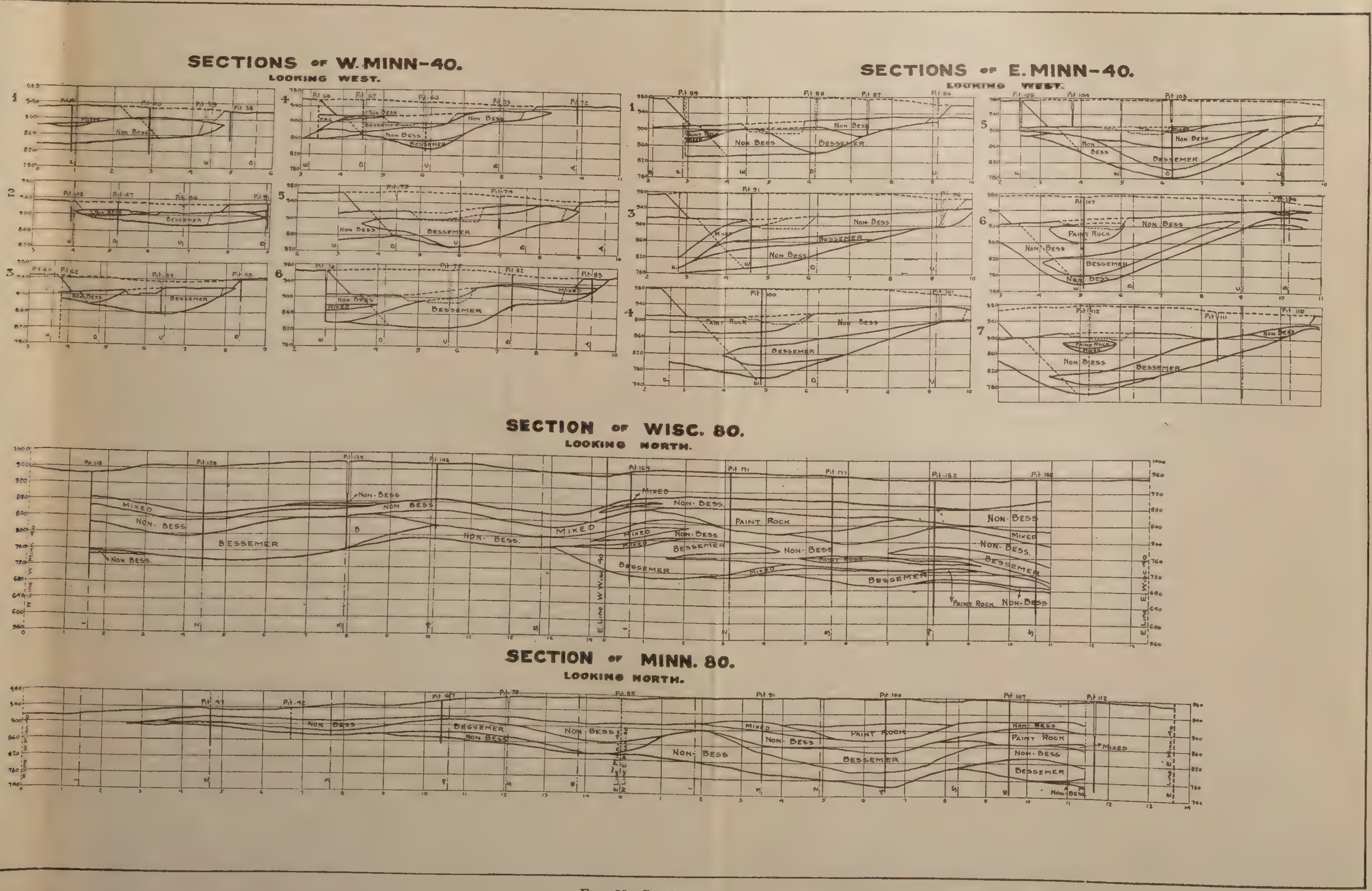


FIG. 82—Sections

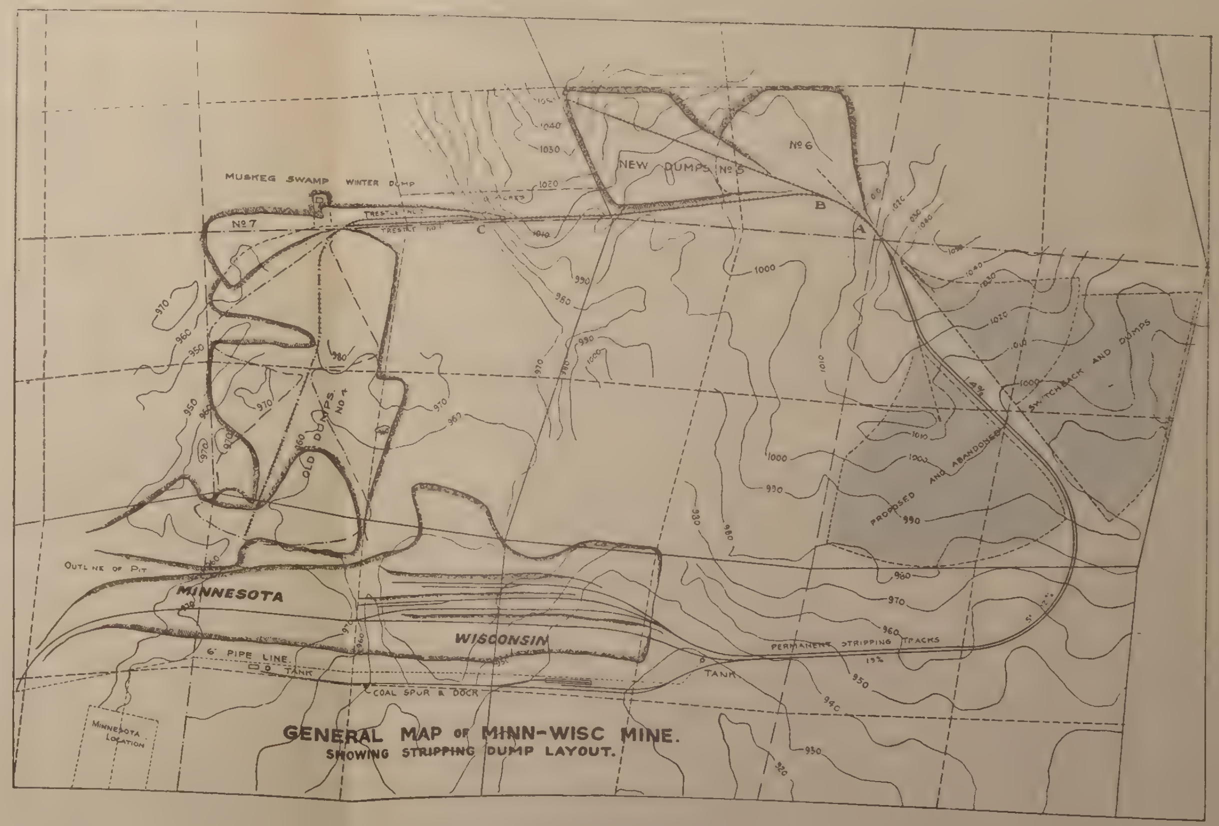


FIG. 83—General Map Showing Dump Layout

pleted No. 4 dumps, the original dumps north of the Minnesota. This was extended west and thrown south and shows on plate as No. 7. From this point a new trestle was run south over the location of trestles Nos. 1 and 2, making in effect the third deck.

Thus dump room was found to accommodate the strippings from the Wisconsin which aggregate over 7,000,000 yards. Stripping on this property commenced in April, 1908. The first or south bench strippings were removed along track No. 1 eastward through Minnesota ground. The second and third benches stripping trains were run out of the northeast corner of the Wisconsin on to dump No. 4. Before the end of 1908 dumps Nos. 5 and 6 were started. On clean-up work switchbacks and steep grades with pushers were freely used. A steep 6 per cent down-grade short cut was used for return of empty train into pit. Approximately 1,500,000 yards were removed during the 9 months of 1908, nearly 2,500,000 in 1909, and the same volume in 1910. From 2 to 2½ tons of ore to the yard of stripping will be exposed upon completion of this work. The pit equipment at this property consisted of:

6 Model 91 Marion shovels, of which 3 to 4 were in use
8 17 by 24 locomotives
6 19 by 26 locomotives
1 100-ton wrecking crane
180 7-yard cars

The aggregate trackage in the pit is approximately 5 miles. The run to the dumps is about 4 miles. The stripping trains make on an average 5 to 6 trips in 10 hours. The actual net running time for a round trip is 50 minutes. The small engines pull 14 cars; the large engines, 18 cars. The crew when operating 3 shovels, 2 shifts, ranged from 350 to 375 men in 24 hours. The illustrations in Figs. 84, 85, and 86 show the work on this property at varying stages.

BUILDINGS AND PIT EQUIPMENT AT OPEN-PIT MINES

BUILDINGS

1. *Offices, warehouses, and location houses.*—These vary in size and completeness of equipment according to circumstances, such as magnitude of operations, nearness to town, possible affiliation with other operations under large District Organization, and many other considerations. The size of the warehouse and the stock carried depend largely upon the extent to which shovels and rolling stock are repaired locally. Location buildings for housing of officials and men naturally vary greatly. The initial expense under this head may run from $15,000 to $75,000 with from $5,000 to $30,000 additional for warehouse stock.

2. *Boiler house, engine house, coal bins, dock, and trestle.*—The power required will depend upon the drainage situation. The volume of water to be pumped out of the pit may be a negligible quantity or it may run into several hundred gallons a minute. A thousand gallons a minute (1½ million gallons per 24 hours) is not an unusual volume of water for a large pit. Apart from the drainage question there is the water required for the operation of shovels and locomotives. The possibility of

having to pump a large volume of water to quite an elevation against the resistance o a long pipe line calls for a varying boiler capacity. Shop-power, lighting and heatin requirements are also variable. The total investment in a plant under this head including coal-dock, water-tower and pipe at the pit, may run up to $25,000.

3. *Shops.*—The woodworking, blacksmith, and machine shops are frequentl housed in one properly divided building of slow-burning construction. The cost o building varies according to requirements from $2,500 to $15,000. The equipmen may run from $5,000 to $20,000.

The round-house may vary all the way from a board and tar-paper shack to a roomy, well-constructed brick building with turn-tables and pits costing from $5,00 to $10,000.

At one well-equipped and well-managed stripping job where there is a tota yardage of over 10 million, the following shop equipment is taken from the surfac plant inventory.

Shop equipment—

1 18-inch Le Blond engine lathe; 12-foot to 9-foot bed.
1 No. 3 Acme bolt cutter.
1 2-inch Spindle emery grinder.
1 Bignal & Keeler pipe machine, capacity 1 inch to 8 inches.
1 28-inch American Tool Works standard swing drill press.
1 Nile Half Universal radial drill with 5-foot arm, capacity to a 2-inch hole.
1 48 by 6-inch cast iron grindstone trough.
1 30-inch lathe, 18-foot to 9-inch bed.
1 100-ton hydraulic wheel press.
1 24-inch Ohio planer, 8-foot to 0-inch bed.
1 1,100-pound Niles steam hammer, single frame.
1 Stewart No. 4 left hand horizontal discharge pressure blower, 30-inch blas wheel.
3 54 by 54 by 24-inch square-base stationary forges.
1 Cleveland combined punch and shear, 26-inch throat.
1 Clement 16-inch planer and joiner.
1 Clement 16-inch band saw.
1 18-inch vertical boring machine, adjustable table; capacity up to 2 inches
1 Greenlee 30-inch heavy up-saw table with adjustable table.
1 heavy power-driven hack-saw; capacity to cut up to 7 inches, fitted wit swivel vise for cutting on the diagonal.

Portable tools and tool equipment—

1 Chicago pneumatic air drill; capacity to 2½-inch hole.
1 Murphy pneumatic air drill.
1 Boyer long-stroke riveting machine. Hammer.
2 Norton 25-ton ball-bearing jacks.
1 Dudgeon 20-ton hydraulic jack.
5 35-ton Barrett jacks.

PIT EQUIPMENT

The pit equipment for a standard stripping job of any magnitude will include:

1. *Steam shovels.*—From 3 to 6 standard steam shovels of the 100-ton type, 3-yard dipper, costing on an average $12,500 each. Cost, $40,000 to $80,000.

2. *Locomotives.*—Two sizes in general use. 17 by 24 and 19 by 26, according to the loads and gradients, costing $10,500 and $13,000 respectively. There will be from 6 to 10 on the average job, and up to 15 on steep grades with long runs to dumps. Cost, $60,000 to $100,000 to $175,000.

3. *Stripping cars.*—New pattern standard 7-yard cars at a cost of from $500 to $600, or Kelbourne & Jacobs 16-yard cars at $1,750 each. Of the smaller cars the number will vary from 100 to 200 according to the length of the trip to the dump. Cost, $50,000 to $125,000.

4. *Wrecking crane.*—One wrecking crane, $10,000.

5. *Track.* From 3 to 8 miles of track representing a supply investment of $4,000 to $7,000 per mile exclusive of grading and tracklaying.

INVESTMENT

Investment.—The total investment represented by a properly equipped open-pit mine may then be expected to vary between the following limits:

Buildings and plant from	$ 40,000 to	$100,000 to	$160,000
Pit equipment and trackage	150,000 to	200,000 to	275,000
Total investment	$190,000 to	$300,000 to	$435,000

CONTRACTOR'S EQUIPMENT

The regular contractor's equipment consists of 2 to 3 shovels, ranging from 60 to 70 tons and the necessary narrow gauge rolling stock. The original cost of a 2-shovel contractor's equipment may be roughly estimated at $100,000:

2	steam shovels, 60 to 70 tons	$ 19,000
8 to 16	dinkey locomotives, 14 to 26 tons	34,000
60	5-yard stripping cars	24,000
3	miles of narrow gauge track	15,000
	Pumps, boiler, and water tank	1,000
	Pipe line	1,000
	Warehouse, pump house, oil house	1,000
	Camp outfit, 200 men	2,500
	Extra repair parts, portable blacksmith and shop equipment	2,500

Such an equipment can, with 2 shovels working 2 shifts, handle from 80,000 to 125,000 yards a month, under bad and favorable conditions respectively, providing the dumps are reasonably accessible. On 2 shovel work the crew will vary between 200 and 300 men per 24 hours, according to conditions. This equipment is, generally speaking, limited to comparatively small jobs, from 1½ to 2 or perhaps 3 million yards.

Under such conditions it has decided advantages over heavier, standard equipment. It is more movable; its narrow gauge track permits sharper curves and steeper grades. A single shovel cut permits double track with narrow gauge. The shovel reports on a well-managed contractor's job generally show larger net operating time. For heavy work and large jobs standard equipment only will ensure continuous operation.

The monthly operating cost of such an outfit on the basis of 52 operating shifts per month may be estimated at $20,000.

Payroll	$12,000
Coal, at $4.25 a ton, delivered	2,500
Explosives ....$500 to	1,000
Oil, illumination and lubrication	500
Hardware and blacksmith material	400
Shovel and engine repairs, heavy blacksmithing, car repairs, etc.	750
Ties and rail	250
Administration	600
Wear and tear	400
Interest on investment	500
Depreciation, 12 per cent annually	1,000
	$19,900

The above estimate is fairly liberal for the average case. The resulting yardage will vary greatly with conditions. It should not fall far short of 100,000 yards a month under normal average conditions, which would establish a 20 cent actual cost. Considering the chances the contractor takes, the prevailing contract prices of 25 to 32 cents would not leave an excessive margin of profit.

EXAMPLES OF OPEN-PIT MINING

The Biwabik Mine.—The Biwabik Mine is of historic interest in that it is the oldest mine on the Missabe. It is of technical interest because it has been the pioneer in the introduction of the steam shovel, of systematic sampling and mixing at the mine, and of heavy crushing machinery. The original discovery was accidental, an uprooted tree exposed the iron ore. This led to systematic prospecting in 1892. In 1893 a mining company was organized and a shipment of 151,500 tons made that year.

The property consists of 160 acres and the ore-body proper covers approximately 80 acres. The ore-body is within 20 feet of the surface on the north. It dips toward the south at about 10°. The thickness of overburden ranges from 20 to 50 feet with an extreme of 90 feet on the south. The average thickness of the ore is approximately 90 feet. The estimates show a total developed tonnage of 14 million in round numbers, with less than 4 million tons left to mine.

There are distinct layers of Bessemer and non-Bessemer ore. The proportion of

Bessemer ore is about 40 per cent of the total. Patches of almost solid seams of hard ore, several acres in extent, are common occurrences. Intermediate layers of paint-rock and of pure taconite occur, broken up and showing movement. The hardness of the ore is such that machine drilling is necessary. Six Ingersoll drills are used, the air compressor is an Ingersoll-Rand with 18 by 11 by 16-inch air cylinders. Holes of varying depth up to 24 feet are drilled on the bench system.

The ground is so thoroughly sampled and mapped out that the chemist can order the blasting so as to obtain approximately the right mixtures in the resultant ore heaps from the different benches. Check samples are taken in from 4- to 10-car lots. By proper selection a good smelting grade can be maintained. The shipments average 55 per cent iron and 5.5 per cent silica for Bessemer ore, 51.5 per cent iron and 6 per cent silica for non-Bessemer ore. This company was the pioneer in systematic mixing at the mine, beginning this in 1900.

Much of the ore from bank blasting required crushing and in 1901 the company built a crushing plant, consisting of a No. 9 breaker with a 21-inch opening. The use of heavier steam shovels made a larger breaker desirable in order to reduce redrilling and reblasting. The Allis Chalmers Company recently developed the No. 24 breaker with a 48-inch receiving opening which can handle the largest and heaviest rocks that a modern 3-yard shovel will lift. The installation has resulted in a minimum of reblasting. When the present plant was being designed, the relative merits of jaw and gyratory crushers were carefully considered and the gyratory decided on for the following reasons: (1) Continuous crushing action with freedom from shock, and consequent cheaper foundation and less liability of breakage. (2) Large circular receiving opening permitting discharge of an entire carload into the breaker without having chutes leading to it filled with material.

The Biwabik plant has a capacity of over 1,000 tons of ore per hour. It is simply a question of getting cars to the dumping platform fast enough. The No. 24 breaker is driven by belting from a 200-h. p. 3-bearing induction motor running at 570 to 600 r. p. m., carrying a 38 to 28-inch pulley. The ore passes over a grizzly having a width of 14 feet across. The bars are spaced at 2 inches apart and the undersize falls through a 50″ spout to one of the shipping bins. The receiving opening of the crusher is 48 inches clear between crushing surfaces and 125 inches long. The crushed product passes through a 6 by 13-foot revolving screen set at a 1¾-inch incline per foot. The undersize from this 2-inch screen joins the undersize from the grizzly. The oversize ranging from 2 to 5 inches falls into a separate shipping bin. A 30-h. p. back-geared induction motor drives the screen. The steam shovels handle chunks of ore up to 6 to 8 tons in weight and the crusher was especially designed for heavy service. Among these provisions is a well-designed system of oil circulation in all bearings. Power is obtained from an Allis-Chalmers 22 to 36-inch simple engine, direct coupled to a 300 k. w., 3-phase, 60-cycle alternator. The current is generated and distributed to the motor at 2,300 volts. A direct current, type K, generator is used. Another 120 k. w. generating unit furnishes electric light to the village of Biwabik and also operates an 800-gallon pump for the village water-supply. Steam is furnished

from a battery of four 72-inch by 18-foot Northwestern tubular boilers of 150 h. p. each, operated at 100-lb. pressure. A very complete machine shop adds to the efficiency of mine and crusher operation.

The approximate cost of the crushing plant is $70,000 installed, including electrical equipment. The average cost of crushing is about 1½ cents per ton. The plant is operated by 10 men, one foreman who also looks after the electrical equipment, 5 dumpmen who unload and feed, 4 brakemen who look after the discharge.

The Shenango Mine.—The Shenango ore-body includes 80 acres. One forty is strictly an underground proposition, while the eastern forty is practically all stripped. The ore-body on this forty lies under from 100 to 200 feet of drift and has a maximum thickness of 300 feet. By many a forty-acre tract under these conditions would not be considered a stripping proposition. This ore-body, however, is unusually thick and of high grade. After careful consideration it was decided that the elimination of losses incidental to underground mining and consequent certainty of recovery of all the ore, together with the greater flexibility of output possible under open-pit conditions, warranted the removal of some 6,000,000 yards of overburden.

The work was started in 1907 with Lima engines. Practically half a million yards were removed the first year. Later a long approach was cut on a 2 per cent grade and four stripping benches were opened, standard equipment being used. By the extension of the stripping on to adjoining ground about 40 acres were stripped and 80 per cent of the ore area was uncovered. Plate IX, Fig. 88, represents this mine in the winter of 1910-1911. Fig. 89 represents the mine at the opening of the 1912 season.

From two to four shovels have been at work steadily; generally 3 shovels, served by 8 trains of 12 cars, each holding on an average 6.2 yards, were used. The run to the dump is 1½ miles on a 2 per cent grade. Following is a statement of the yardage and ore tonnage to January 1, 1912.

Year	Yards stripped	Tons shipped	Lean ore stockpile
1907	450,405	....	....
1908	1,062,761	203,821	....
1909	1,256,647	453,654	....
1910	1,510,909	602,610	93,423
1911	1,473,864	405,791	537,065
Total to Jan. 1, 1912,	5,754,586	1,665,876	630,488

This ore is to be removed in so far as possible by a system of spiral tracks and switchbacks on a 2.7 per cent compensated grade, over which 5 steel cars can be hauled. Whatever ore is out of reach of this track system, can be removed through the shaft.

Through the kindness of the Shenango management, I am enabled to furnish th following statement of daily stripping operations under varying operating condition averaged over one month.

YARDAGE PER MAN, SHENANGO PIT.

Case	Operating conditions	Number men			Yardage per man		
		Day	Night	Total	Day	Night	Total
A	2-shovel stripping job in coldest weather under poorest possible conditions....	146.70	58.77	205.5	8.31	21.47	12.08
B	3-shovel job, good conditions, winter weather	188.75	78.03	266.78	14.11	35.34	20.1
C	3-shovel job, summer weather, good conditions. Low average due to fact that much of the stripping was "clean-up" work	192.55	78.29	270.84	14.98	35.24	20.81

CLASSIFIED LABOR STATEMENT ACCOMPANYING ABOVE AVERAGES.

	Case A			Case B			Case C		
	Day	Night	Total	Day	Night	Total	Day	Night	Total
STEAM SHOVEL—									
Engineers	2	2	4	3	3	6	3.2	3.1	6.3
Craners	2	2	4	3	3	6	3.1	3.2	6.3
Firemen	2	2	4	3	3	6	3	3	6
Pitmen	8.4	8	16.4	12	12	24	15.1	15.5	30.6
LOCOMOTIVE—									
Engineers	8	8	16	10	10	20	9	9	18
Firemen	8	8	16	10	10	20	9	10	19
Brakemen	2	2	4	3	3	6	3	3	6
Switchmen	4	6	10	6	7	13	7	8	15
GENERAL—									
Foremen	1	1	2	1	1	2	1	1	2
Track bosses	2.2	..	2.2	4	..	4	2	..	2
Trackmen	30.4	2	32.4	40	4	44	45.5	3	48.5
Drillers and blasters..	23.2	1	24.2	28	1	29	14	1	15
Timekeepers	1	1	2	1	1	2	1	1	2
Dump bosses	3	2.1	5.1	4	3	7	3	2	5
Dumpmen	28	12.6	40.6	29	14	43	47	15	62
Extra (Salt, Etc.)....	3	3	6	3	3	6	18	1	19
Shop labor, chargeable to stripping	27.9	..	27.9	22	..	22	15.5	..	15.5

The Stevenson Mine.—The Stevenson operates a long, narrow fairly deep pi about one mile in length. Width on top of pit ranges between 450 and 900 feet, deptl of overburden 50 to 55 feet. The thickness of ore is very irregular, 10 to 100 feet The unevenness of the top of the ore-body necessitated much hand cleaning, an added to the expense of stripping. The stripping work on this property was largel contract and cost between 30 and 32 cents a yard.

On the bottom and on the south side of the ore-body the ore is interbanded wit layers of taconite from a few inches to 10 feet thick. Quite a force of men i employed at each steam shovel sorting out rock. Many large horses of taconite (se Fig. 91) project into the ore. Some are as much as 600 feet long, 75 feet wide, an

FIG. 90.

General View of Stevenson Pit. The cross indicates the approximate location of the "horse" and the hand cleaning shown in Figures 91, 94, 95.

60 feet high. They are avoided as much as possible. Often they are not exposed by the drill holes and are unavoidable, in which case they add to the expense by necessitating much sorting. The maximum number of men employed on pit work is about 350; from 35 to 40 per cent of the maximum crew is employed in sorting. Figs. 90 to 96 fully illustrate the conditions at this property.

The output varies with circumstances and market conditions from ½ million to 1¾ million tons of a very fair grade of ore. The cost of this ore on the cars at the mine is a trifle over 50 cents a ton, including therein a flat charge of 20 cents a ton for stripping. This figure is based on at least a ¾ million output for the season.

FIG. 91. Formation at Stevenson Mine, showing large "horse."

This company is operating quite a distance from town and has a well-equipped machine shop which represents an investment in building and equipment of some $30,000. It costs between $30,000 and $40,000 a year in labor, power, and supplies to operate this shop which meets both pit and underground requirements.

FIG. 92. Formation at Stevenson Mine, South Rim.

FIG. 93. Formation at Stevenson Mine, North Rim. Showing suspension bridge across the pit.

FIGS. 94 and 95. Hand Cleaning Top of Ore.

COST OF OPEN-PIT ORE

The cost of open-pit ore is a subject upon which it is difficult to generalize. To begin with, the amount of overburden and the cost of stripping per yard are both very variable. Stripping costs vary from 40 cents to 12 cents a yard, according to the size of the job, the operating conditions, and the equipment used. Contract prices vary from 25 to 32 cents on contracts of 500,000 yards or over. The writer has seen company cost-sheets on big work where, including depreciation and repairs, the costs have ranged from 14 to 20 cents per yard under favorable conditions and from 16 to 24 cents under decidedly unfavorable conditions. Many jobs report higher costs. The capital invested in stripping and the interest charges thereon vary between wide limits. A 40-acre tract may require stripping to a depth of 100 feet—a 1½ to 1¾ million dollar stripping investment, and perhaps 3 years to complete it in. The annual interest charge on this alone would be $100,000. If this is charged against 8 million tons of ore to be mined at the rate of a million tons a year, the stripping charge against the ore would be, roughly, 30 cents per ton of ore, to which must be added the cost of mining. On the other hand, there are large open-pits whose cost of ore on the cars, including mining, stripping, and local overhead charges, runs from 15 to 18 cents per ton.

The cost of mining varies principally with the hardness of the ore, the amount of rock sorting needed, regularity of iron and phosphorus content, and the transportation question. It may be said that the cost of open-pit ore on the cars, including all local charges, but excluding royalty (or interest on fee investment), taxes, and extraneous overhead charges, may vary from 15 to 75 cents a ton under extremely favorable and unfavorable conditions respectively. Excluding the especially favorably situated Mahoning-Hull-Rust ore-body, which is in a class by itself, it may be said that the average cost of open-pit ore in the central Missabe will fall within 40 cents on the cars under ordinary and within 30 cents under favorable operating conditions.

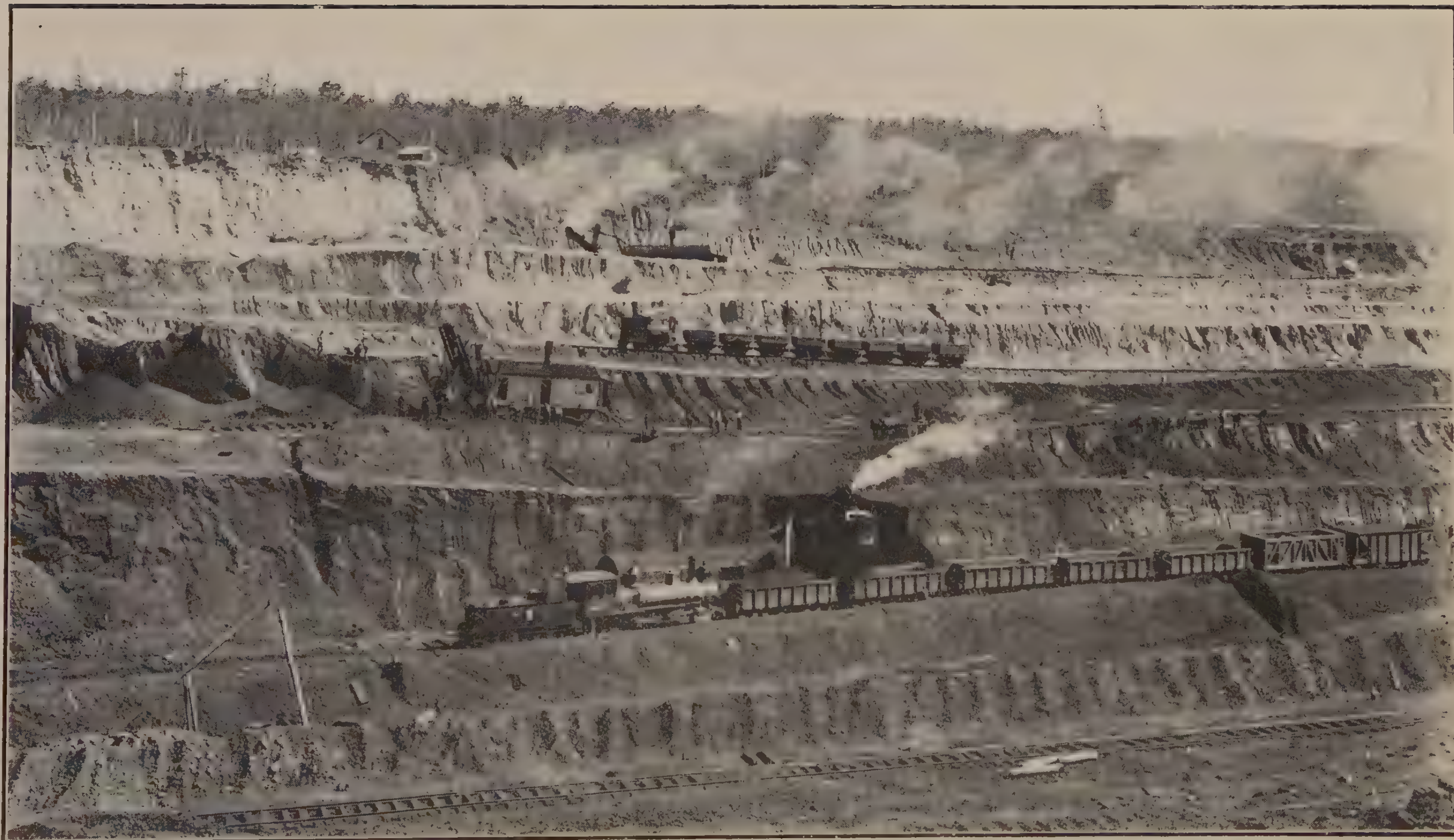


FIG. 96. Stripping and Mining Ore, Stevenson Pit.

THE WESTERN MISSABE RANGE

Some of the earliest exploration work on the Missabe was done in the stretch of country beginning at the village of Nashwauk in Itasca County and extending southwesterly across the Mississippi River into Cass County, a total distance of 35 miles. This area is to-day known as the Western Missabe district.

Historical.—As early as 1810 Major Pike visited Pokegama Falls on a trip to the headwaters of the Mississippi. Lieut. Allen, H. R. Schoolcraft, J. N. Nicollet, and others visited the Falls in the early '30's and '40's. In 1866 Henry H. Eames, the first State Geologist of Minnesota, reported iron ore at Prairie River Falls and stated that the Missabe Range contained immense bodies of iron ore. In 1880 the U. S. Geological Survey sampled the outcropping iron formation and in 1883 Geo. A. Fay and Frank Mills bought the first lands purchased in Itasca County on the Missabe Range for mineral purposes. In 1886 the North Star Mining Company was formed to explore along the entire length of the Range. A little high-grade and much sandy ore was found in the early '90's.

Prospecting by test pits and diamond drills disclosed ore analyzing 50 to 63 per cent iron on the Diamond Lands in Secs. 15, 56, 24, and a news item in the Pioneer Press of 1889 mentioned open-pit development as a likely method of mining this deposit. Subsequent explorations showed good ore intermixed with layers of sandy ore. In 1892 prospecting was active at many points. Work was begun on the Arcturus in Secs. 13 and 24, 56-24. Later work on this property developed the wash ore characteristic of this district. The Missabe Chief was also being developed in Sec. 23, 57-22, and with such success that in 1893 George A. Fay reported merchantable ore developed.

The panic of 1893 caused a cessation to exploration work in the district. No great bodies of high grade ore had been found. Some good ore was found, but nearly all the explorations reported layers of sandy ores intermingled with the better ore. When active work was resumed on the Missabe, attention was directed to the eastern part where large bodies of high grade ore had been found.

In 1901 a carload of ore from the Arcturus was shipped to Georgia to be treated in a log-washer. Results were satisfactory. In 1902 an experimental washing plant was built at the mine. A large body of sandy ore had been developed. Considerable money was spent in development, washing experiments, and attempts to interest capital in the property. It was clear that this was a problem calling for unlimited capital and the option was finally dropped.

In 1904 the Oliver Iron Mining Company secured an option on the property and at expiration of the option took a lease. Convinced of the feasibility of successfully

IDEAL CROSS-SECTION OF A WESTERN MISSABE RANGE ORE-BODY

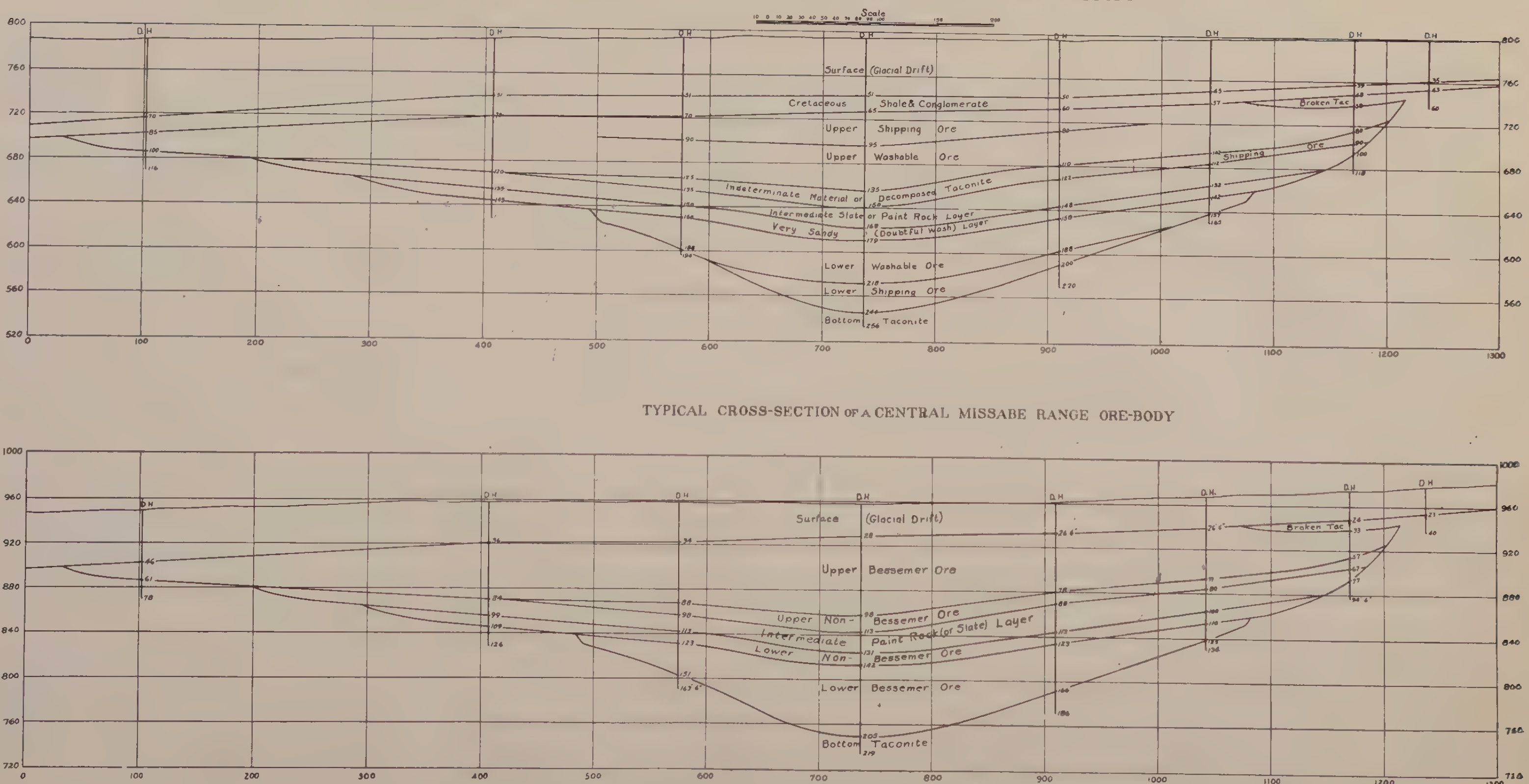


FIG. 97. Cross-Sections.

handling the ore, they greatly increased their holdings in the district. In 1906 the D. M. & N. Ry. was extended to Trout Lake and the work of stripping and developing the Canisteo ore-bodies was begun. The work was laid out on a large scale. Exhaustive experiments were made to find a suitable treatment for these ores. Finally a mill was erected whose equal in production is not to be found in the United States to-day. Civic and municipal development went hand in hand with mining development. Coleraine, built at the north end of Trout Lake, is a model city.

IMPORTANCE OF THE WESTERN MISSABE FROM AN ECONOMIC AND CONSERVATION STANDPOINT

As development was progressing on the Canisteo and other properties, the new unexplored territory under lease was being drilled, and the ore reserves of the district were rapidly increased. Some large bodies of high grade ore were developed. With these occur layers and bodies of lean or "wash" ore. As this lean ore must be removed in order to mine the merchantable ore, it was highly important both from the standpoint of economy of operation and the conservation of national resources to find a method of concentrating this into a merchantable product. The Oliver Iron Mining Company undertook the task of solving the concentrating problem. They constructed a small experimental mill where extensive experiments were made during 1907 and 1908. The splendid $1,500,000 mill at Coleraine is an enduring monument to their pioneer work in this line.

Western Missabe ore-bodies.—The ore-bodies of this district resemble those of the central and eastern parts of the Missabe Range in shape, size, and general structure. There is, however, a very important difference in the character of the ore. For purposes of comparison the western deposits have been called Washable-Ore bodies, and the central and eastern, Standard-Ore bodies. The latter ores are merchantable, can be shipped direct. Western Missabe or Washable-Ores must be treated in order to make them merchantable.

Typical washable-ore is composed of alternating layers of high grade ore and a free, fine sand, almost an impalpable powder. The mass is easily broken up in mining, making a mixture of small chunks of hard ore and very fine sand. This ore lies in certain definite layers and there are all gradations from it into merchantable ore, very sandy, lean ore, broken and decomposed taconite, and solid taconite. The decomposed taconite may or may not be washable; actual trial is generally necessary to determine. The washable kind consists of chunks of fairly good ore with fine sand and considerable granular iron; the sand is not as free nor in as definite layers as in the typical washable ore.

Though the classes of ore above enumerated seem upon superficial examination to be indiscriminately intermingled in these Washable-Ore bodies, there is a definite relation between them and the Standard-Ore bodies of the central and eastern part of the Range. In the latter the paint-rock or slate layer divides the ore-body into an upper and a lower high grade body, each having immediately adjacent to the paint-rock a thin layer of medium grade non-Bessemer ore. (See cross-sections, Fig. 97.) In the washable bodies the main high grade layers are represented by merchant-

able ore and the typical washable ore (the lower body being the more typical), anc the non-Bessemer layers are represented by decomposed taconite (above paint-rock) and very sandy, lean, washable ore (below paint-rock), both classified as "Indetermi nate Material," because from the drill records or inspection of ore in the ground the can be classified as neither "Shipping" nor "Washable" ore. Actual trial is generall necessary to determine the merchantability of products made by washing such ores The two cross-sections referred to show the analogy between the two classes of ore bodies.

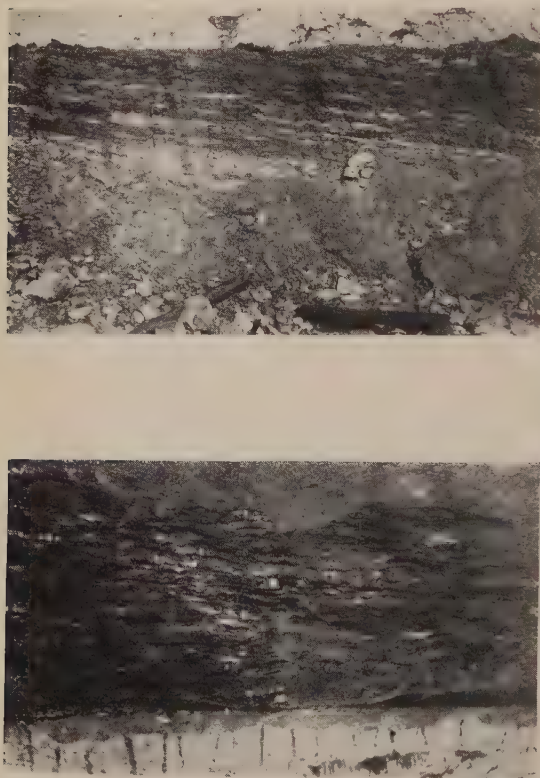


FIG. 98

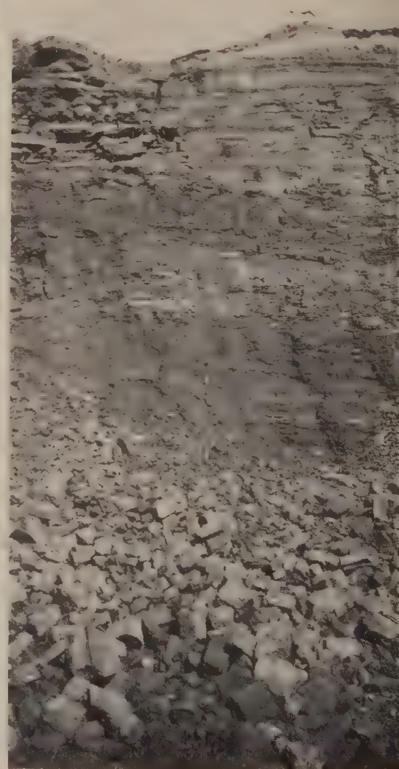


FIG 99.

Typical Washable-Ore Banks.

While these general divisions can be made in every Western Missabe body, thes layers are not always persistent in their typical development. The shipping- o mechantable-ore grades into washable-ore, the latter into broken and decompose taconite, and the decomposed taconite ("Indeterminate" layer) into merchantable-or or solid taconite. In general there is a much greater prevalence of layers of soli and broken taconite in the Western Missabe ore-bodies than in those of the centra and eastern part of the Range. The illustrations in Fig. 98 show a bank of typica

Upper Washable-Ore, and beneath it typical Lower Washable-Ore. The latter shows clearly the layers of hard ore and interbanded seams of white sand much of which has been blown out by wind or washed out by rain. The right hand view, Fig. 99, shows a bank of typical Upper Indeterminate Material, characterized by large chunks and absence of definite layers of free sand. Overlying every ore-body of the Western Missabe wholly or in part is a layer of detrital material 5 to 20 feet thick, which is better developed here than at any other place on the Range. At its contact with the iron formation proper is a coarse conglomerate of iron ore pebbles about a foot thick grading upward through a finer conglomerate into a very fine grained black shale. In places it contains as much as 50 per cent metallic iron, but it will not "wash." Shells and other organic remains are found in it. Teeth and vertebrae from it have been recognized as those of the Mosasaur of the Cretaceous age and therefore this formation is known as the Cretaceous Shale and Conglomerate. Because of organic remains contained in this overburden it is noticeably high in phosphorus. The ground-waters have carried this phosphorus down into the iron ore proper, fairly saturating the upper 10 or 15 feet of the ore-bodies with it.

MILL PROCESSES

The first experiment in the treatment of these sandy ores was a test made in a log-washer at Cedartown, Ga., in 1901. This showed a concentration of 3 tons into 2 and a grade of over 60 per cent iron, proving conclusively that a merchantable product could be made. In 1902 an experimental plant was built at the Arcturus Mine, consisting of simple revolving screens and log-washers. The results corroborated at log-washer test but emphasized the necessity of large scale operations. In 1903 and 1904 an experimental plant having revolving screens was built at the Holman Mine. To these early experimentors belongs the credit for the pioneer work of determining the essential fact that it was possible to separate the sand and the large pieces of ore. The recovery of the granular ore from the fines was a minor question for subsequent detailed investigation.

In 1907 the Oliver Iron Mining Company built an experimental mill at Coleraine. During the seasons of 1907 and 1908 large scale, detailed experiments were carried on and various machines were tried for the treatment of the fines or slimes overflowing from the large log-washer. Finally a small log-washer, named "turbo," was developed for these slimes which were further treated on Overstrom tables. From the results of these experiments a permanent mill was designed of which a brief description follows.

Trout Lake concentrator.—The plant is situated on the east side of Trout Lake, readily accessible from all directions. It was designed by the engineers of the Oliver Iron Mining Company and erected by the American Bridge Company. The equipment was furnished by the Allis-Chalmers Company of Milwaukee. The exterior view of the mill (see Fig. 101) taken from the northwest corner shows the table addition and tailings lander. The concentrate bins and tracks are on the opposite side. The mill building is of heavy steel construction throughout, 255 feet long, 162 feet wide, and

124 feet high, enclosed with corrugated iron. The approach to the mill is an enormous earth fill, some 4,000 feet long, containing several million cubic yards of strippings from the Canisteo and Walker pits. It has a maximum height of 125 feet and was planned to accommodate four tracks. A steel trestle 650 feet long connects it with the mill. At the opposite end of the mill 300 feet of additional steel work is in place, and is now being used for tail track; it can be utilized for an addition to the mill if need arise.

The power and water for the mill are supplied from a power and pumping plant 7,000 feet distant on the shore of Trout Lake. The pump has a capacity of 500,000 gallons per hour. The water is pumped direct to a 100,000-gallon supply tank at the mill. All the machinery in the mill is electrically driven. A 1,250 k. v. a. direct-connected generator transmits electric power at 6,600 volts to the mill where it is stepped down to 440 volts.

The total amount of steel in the mill building, approach, tail-track, and power-plant structures is approximately 7,000 tons. The total cost of mill and equipment according to the published report of the United States Steel Corporation approximates $1,500,000.

The concentrating machinery is arranged in five units, each complete and capable of independent operation. This was done in order to keep the machines within reasonable size, to be able to handle separately the ores from different properties at the same time, and to increase the tonnage. The tail-track already mentioned is long enough for seven additional units. The crude ore from the different mines is hauled to the mill over the big fill approach and trestle and dumped into the bins at the top of the mill. Each unit has a separate bin of 450 to 500 tons capacity. The ore is sluiced out from the bottoms of the bins by a hydraulic jet, and descends by gravity through the different machines. The crude-ore tracks are 90 feet above the concentrate tracks. The ore is handled entirely by gravity; there is no elevating machinery to get out of order, other than the sand pumps necessary to lift the table concentrates to dewatering tanks over the concentrate bins.

Each unit consists of the following equipment listed in the order in which the crude ore passes through it. Reference to the unit flow-sheet (Fig. 100) will help to an understanding of the process.

One crude ore bin and one bar-grizzly.

One 20-foot conical revolving trommel with 2-inch plate openings.

One picking-belt, to receive and convey the trommel oversize to the concentrate bin. (The taconite chunks are picked off the belt and dropped into a rock-bin.)

Two 25-foot log-washers, one on either side of the trommel to treat the trommel "throughs," discharging a product into the concentrate bin.

Two chip-screens, one for either "log," receiving "log" overflow.

Two settling tanks, receiving chip-screen "throughs."

Two 18-foot "turbos" (small log-washers to treat "fines"), receiving the tank settlings, discharging a concentrate directly into concentrate bin.

Four settling tanks receiving the overflow from turbo and first settling-tanks.

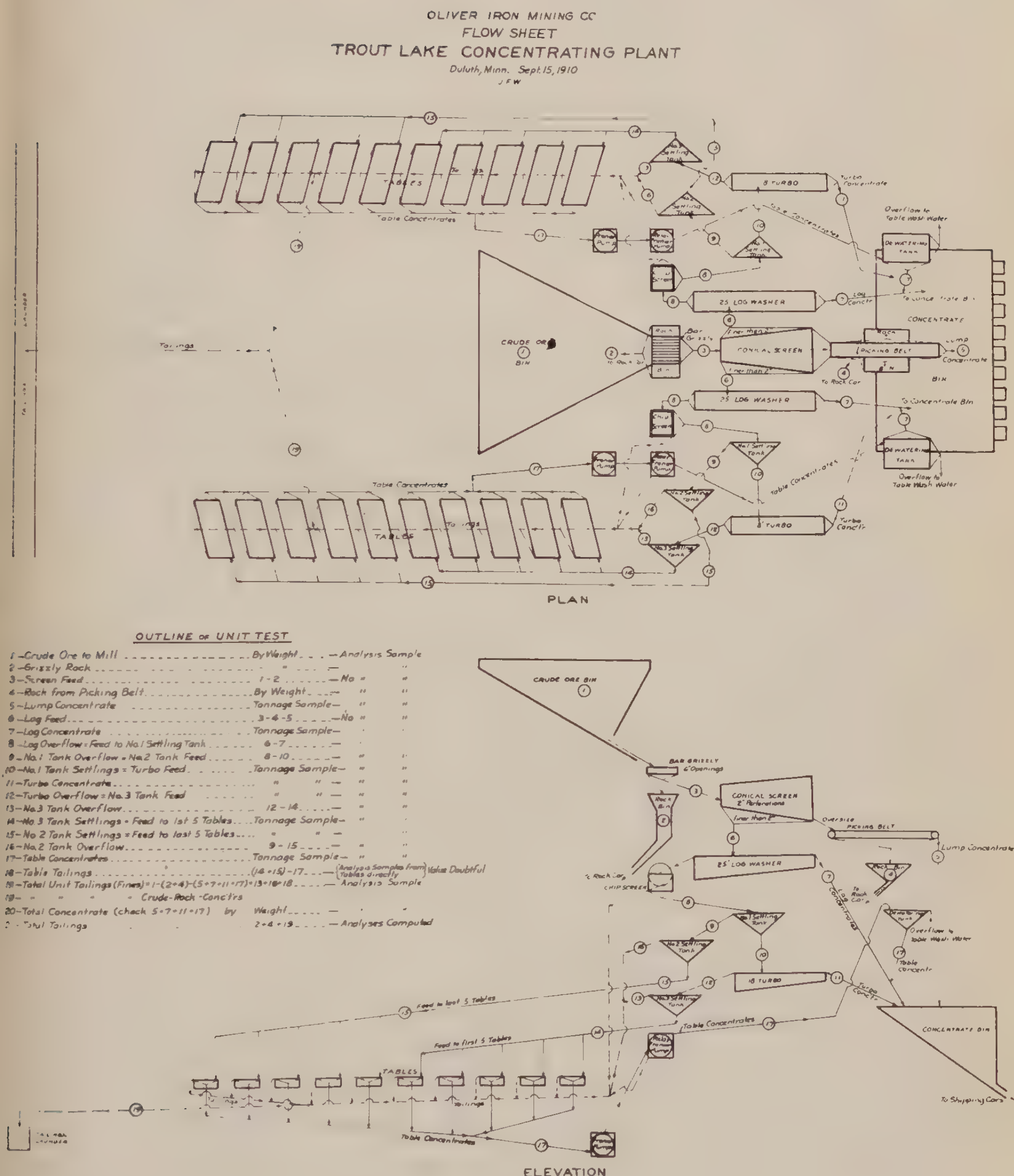


FIG. 100. Flow Sheet.

Twenty Overstrom tables arranged in 2 rows of 10 each, to treat the settlings from the four settling tanks just mentioned.

Eight Frenier Spiral sand-pumps (four primary and four relay), to pump the table concentrates to dewatering tanks from which they discharge directly into the concentrate bin.

The mill product thus consists of belt product, log-, turbo-, and table-concentrates. Each unit has its own concentrate bin of 90 tons capacity. Tracks for ore cars run directly under the bins, a separate track serving each bin. The cars are stored in a yard south of the mill from which they can be spotted under the bins by releasing the brakes, as the grade from the yard is down toward the mill. The tailings consist of chip-screen discharge, No. 2 and No. 3 settling-tank overflow and table tailings. They are collected by launders in the mill basement and discharged into a concrete launder outside the table addition to mill. This launder discharges into Trout Lake, some 2,000 feet distant. The amount of water used per unit is 1,300 to 1,500 gallons per minute, or 78,000 to 90,000 gallons per hour, approximately 900 gallons per ton of concentrates.

The rock picked off the belt and bar-grizzly is drawn from the rock-pockets or bins into a rock car and hauled by an electric motor to a stock-pile east of the mill. One motor and car serves all five units. One 100-h. p. electric motor drives the trommel, picking belt, logs and turbos; one 15-h. p. motor drives the tables and chip-screens; one 20-h. p. motor runs the sand-pumps in the basement of the mill.

The mill building is very commodious and the machinery is conveniently arranged. Safety devices, machine guards, and protecting railings are installed throughout for the safety of employees. The two interior views of the mill were taken early in 1910 before unit 5 and all the safety devices were installed. Fig. 102 shows the trommels and Fig. 103 the table-floor.

In 1910 the mill was operated with two eleven-hour shifts. In 1911 this was cut down to two ten-hour shifts. The average capacity exceeds 200 tons of crude ore per hour per unit, the total for the season of six months being about 3,000,000 tons crude ore. The mill can not be operated in freezing weather, so its operating season coincides with the ore-shipping season. Nearly 100 men are employed on the day shift and 75 to 80 on the night shift.

The average grade of crude ore varies greatly, depending on the character of material being treated and the local conditions at the time of mining. The concentrate product varies within wide limits, depending upon the character and class of ore, just as the grade of "direct-shipping" ores varies greatly. This is not a matter of degree of concentration produced by the mill, but rather the amount of concentration and enrichment produced by nature.

Results.—The operating company refuse to give definite information regarding mill efficiency and tonnage recovery. The results obtained vary widely for different ore-bodies and even for different layers in the same ore-body. The quantity of rock sorted out in the pit is so variable that the mere statement of mill recovery might give a very misleading idea of the total percentage of recovery from these low grade

ore-bodies. The following statement must therefore be taken as an approximation of the results attained and not as authentic first-hand information emanating from the operating company. "The 5 units produce each from 1,800 to 3,300 tons of concentrates per 24 hours, depending wholly on the nature of the feed. The variation in the latter is so great that it is difficult to make serviceable statements. The percentage of extraction varies from 50 to 60 per cent in the low grade material, to 65 per cent or perhaps better in the higher grade material. To illustrate: It may be said that from 2¾ tons of 35 per cent Fe ore-material, 1 ton of 57 per cent Fe concentrates may be expected, while 1¾ tons of 45 per cent Fe ore-material might be expected to yield

FIG. 101. Trout Lake Concentrator.

1 ton of 57 per cent Fe concentrates. The approximate iron and phosphorus content of the concentrates is, iron 56.5 per cent, phosphorus .060 per cent." The following analysis approximates the shipping product of the Holeman and Hill mines, that is to say, the shipping mixture composed of mill concentrates and selected ore from the pit: 56.81 per cent Fe; .047 per cent P; 11.90 per cent SiO_2; .34 per cent Mn; 10.45 per cent moisture.

It may be authoritatively said that the mill has satisfactorily solved the problem of economical handling of Western Missabe ore-bodies, a most important achievement in the line of conservation of our national resources.

MINING OPERATIONS

The present producing mines in this district are the Canisteo and Walker open pits at Coleraine and Bovey, the Holman pit at Taconite, and the Hill pit at Marble.

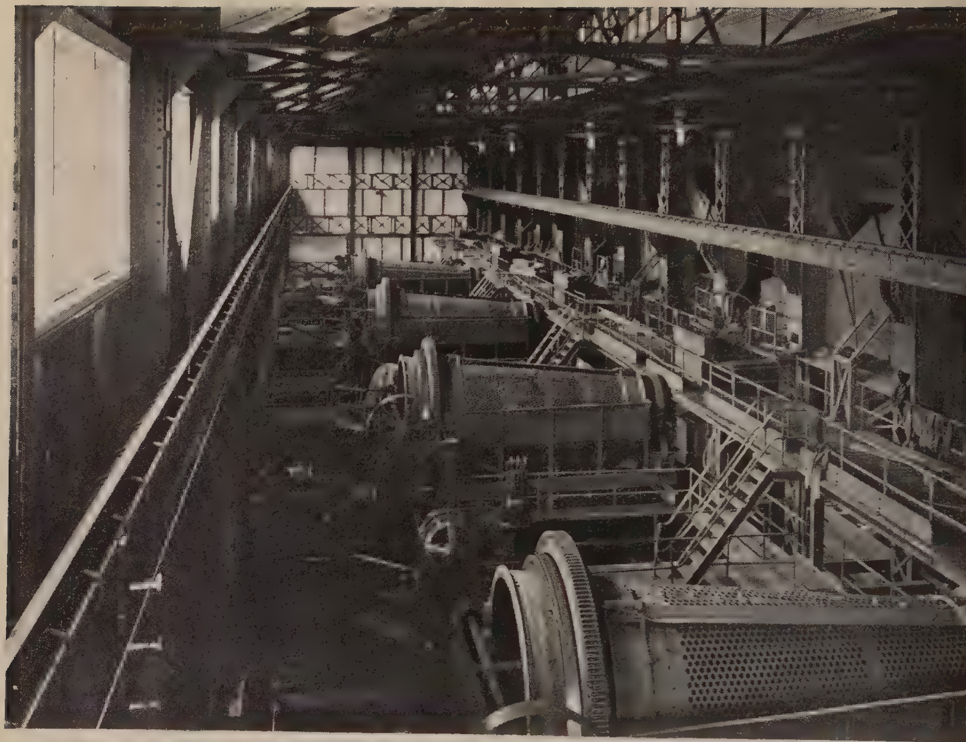


FIG. 102. Trommels.

FIG. 103. Table Floor.

Aside from some underground drifting and raises for drainage and exploration purposes, no underground mining of washable ore is done. The Canisteo and Holman pits were started first in 1906; the Walker was started in 1907, and the Hill in 1908. In 5 years nearly 25,000,000 cubic yards of strippings and 8,000,000 tons of ore have been removed from these pits. Of this ore, 7,500,000 tons were mined in 1910 and 1911. The overburden is quite thick in this district, averaging 80 feet or more, so that thin ore-bodies can not be stripped profitably.

In mining operations frequent test-pits are sunk in the ore, samples are taken of the ore banks, and classifications made of washable and merchantable ore. Whatever ore the ore-grader can use without concentrating, either by itself or to mix with other ores at the docks, is shipped direct. All other material of washable character is shipped to the mill. Solid and broken taconite, paint-rock, and cretaceous shale can not be washed.

The immense size of the open pits, the large quantity of ore already proven up, the great outlay for development, mining and transportation equipment, the capacity, simplicity and operating economy of this concentrating plant, and the size and activity of its cities mark this district as one of the most important on the Missabe Range.

VERMILION RANGE MINING METHODS

ELY MINING DISTRICT

This district covers about one square mile and comprises a group of 5 producin mines, leased and operated by the Oliver Iron Mining Company. The first out-cro of ore was found in 1883 and the first shipment was made in 1888 with a total o 54,612 tons for that season.

The "Ely Trough" is a lenticular iron formation included between walls o greenstone, overlaid by a jasper capping. The structure is synclinal, both longitudi nally and transversely. The mines named in order from west to east are the Chandler Pioneer, Zenith, Sibley, and Savoy. Transverse sections, especially at the wester end on Chandler-Pioneer ground are U-shaped. At the Zenith about midway in th trough an anticline separates the Zenith ore-body into two portions, one upon eithe limb. This same anticline has separated the trough longitudinally into two synclines one between the Chandler-Pioneer ore-body and the Zenith, the other between th Zenith and the Savoy.

The iron formation varies in width from 150 feet to 1,400 feet. The ore lie at or near the bottom of the trough at contact with the Ely greenstone. It come to the surface only at one point, at the west end of the Chandler. The trough has steep easterly pitch and the ore-body rapidly disappears under the jasper capping Exploration work in the district is carried on by means of diamond drilling, th jasper capping being from 300 to 1,000 feet thick.

The Chandler-Pioneer ore-body is the largest in the district. Shipments to dat from these two mines have aggregated over 17½ million tons. The Chandler wa for several years supposedly worked out. Recently some new ore was discovered fron which over 100,000 tons have already been shipped. The Pioneer Mine has an esti mated reserve of at least 4½ million tons with probabilities of developing more or in the future. The mining method used at the Pioneer Mine is typical of the district Two idealized sections (Figs. 104, 105, Plate X) show the general form of th ore-body and the method of development.

"A" shaft on the north was sunk in the greenstone near the edge of the ore-body An unsuspected ore-body was subsequently found near the surface and very close t the shaft. This caused the upper portion of the shaft to swing about four inches ou of line. From this shaft levels are run at elevations indicated on the geological sec tions. The present method is to cut stations and run levels at 100-foot intervals Each 100-foot lift is cut up by two sub-levels or "subs" into three backs of 33⅓

feet each, numbered as shown on Fig. 106. The former practice (abandoned on the 11th level) was to run 3 subs and divide the lift into four 25-foot backs.

When the shaft reaches the 100-foot point below the last level, the new station is cut and a straight drift is run out to the ore parallel with the long dimension of the shaft. The timber station is cut (width about equal to length of shaft) and the timber track is run to join the ore track as shown in the plan of the Pioneer 11th and 12th levels. After the ore has been reached the level is laid out in so far as possible to conform to the plan made in the engineer's office, based on the development of the level above. The level is subsequently cut up into a series of pillars as regular as the ore contours and rock conditions permit. The fee-owners insist on clean mining and their representatives keep a strict watch. This plan of working gives a regular succession of similar levels, fairly regular blocks, and but few irregular bunches of ore that must be mined at extra expense by top-slicing methods.

As the main level advances vertical raises are started at 25-foot intervals along the main level workings. From these raises, the first subdrifts are extended just 33⅓ feet above the main level track. The subs are run parallel to main level openings. They are not run immediately over the latter, but are offset from one-half to a full set. The reason for this lies in the fact that the main level raises from which these subs run are started so that the wall of the level is coincident with the axis of the raise, i. e., the latter is half in the wall and half in the level.

As the first sub-level openings are advanced sufficiently to afford outlets and the safety of miners is assured, the vertical raises are extended 33⅓ feet to the next sub, and the second sub-drifts similarly started. In the subs the chutes are so close and the floor is so uneven that tramming is neither necessary nor advisable. Whatever ore will not "run," is handled in ordinary wheelbarrows.

To understand the mining method in detail, consider the ore-body caved down to the first sub of the 11th level. The 12th level and both its subs are opened up and the mine is in shape to commence drawing the 11th level main pillars which have stood as a protection for this level during the mining of the corresponding 10th level pillars and the 2d and 1st subs of the 11th levels. These pillars are therefore more or less crushed and quite ready to cave if given a chance. The tracks on the 11th level are joined together and the level is cut up into smaller pillars 25 feet square preliminary to caving.

Caving. —Caving is always begun at the extremity of the ore-body furthest from the shaft as follows: The drift set in which a raise is located is extended one or two sets in the direction of the wall and also both ways parallel with the wall. A cave is started by blasting down a pear-shaped opening with ordinary pop shots (see Fig. 112). The jumper drill is then used to enlarge it. This ore must be shoveled into the raises. As the pear- or dome-shaped opening enlarges, it becomes heavy, the ore ultimately crushes and runs into the raises. This is called a "chance." Above the ore zone we have the timber-mat and crushed jasper from the old workings and the "chance" runs out when the timber-mat appears. Two men operate a chance; they pry down with a long bar to make the working safe and then climb into the dome and

put in another series of pop shots. The raise is covered with a log grizzly. The o usually breaks quite fine. The larger chunks of jasper are caught and thrown asid Sometimes large masses fall that need buldozing. When a chance runs out, i. e when the timber-mat appears, the opening is crushed or lagged up; another chance started in a different direction by removing the lagging and repeating the operation o a new face. We have assumed the mine wholly caved to the bottom sub of the 11t level. As a matter of fact such a condition never occurs for operations on the 11t level pillar are going on at the extremity of the level coincident with caving in th subs above. Similarly caving may be in progress on the second sub of the 12th lev at the point furthest from the shaft, while the 11th level pillar is being caved an

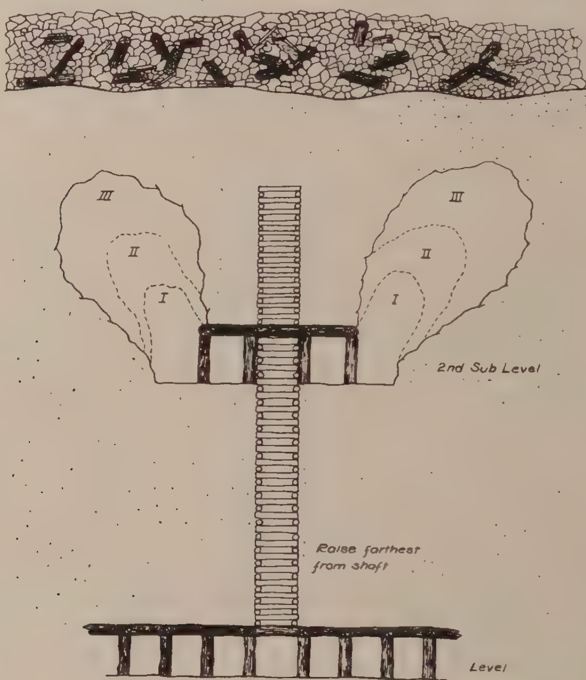


FIG. 112. Starting a Cave.

drawn nearer the shaft. It must be understood that caving is carried on at severa horizons and that the work on the upper horizon is always well in advance of th work on the horizon below so that each working place has above its proper ore bac the caved timber mat and jasper. Fig. 106 illustrates the mine in process of cavin between the 11th and 12th levels. The 11th level pillars are not wholly removed an yet caving is being carried on over the 1st sub of the 12th level.

In caving any square block, even with several vertical and inclined chutes aroun it and in it, a pyramid of ore remains untouched. These "hog backs," as they ar locally termed, are tapped by inclined raises *A* and *B* from the sub below. Some

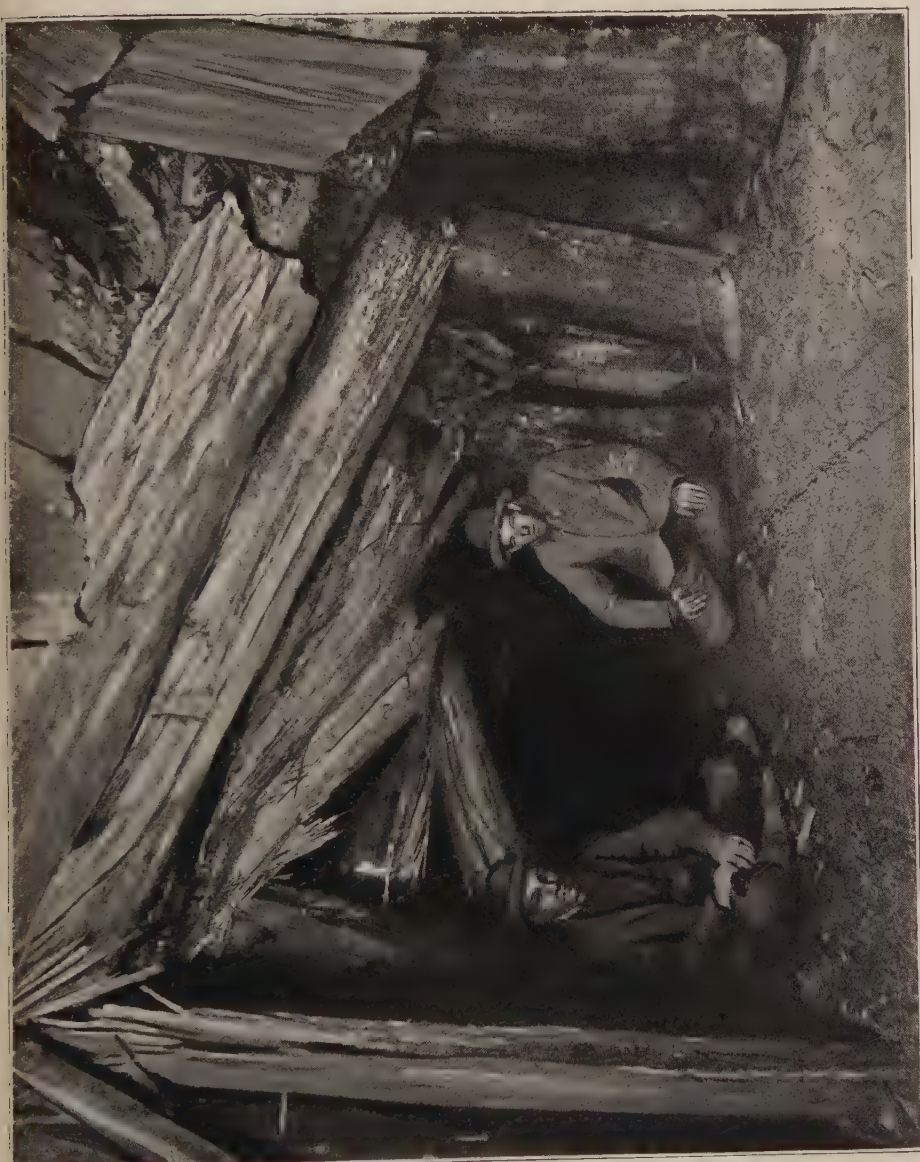


FIG. 113. Drift Timbers Crushed by Heavy Caving Ground.

times a drift is run into one of these, the ore pulled down on the retreating plan an wheeled to the chutes. A complete cycle in the development of this mine might b given as follows:

The 11th level pillar in process of drawing. The 12th level subs completed an both subs in process of drawing. The 13th level in process of blocking out and it first sub also being blocked out but in lesser degree. The 14th level opened u simply by haulage drift into the ore for drainage purposes. The shaft in process o sinking (see Figs. 107 to 111, Plate X). This cycle covers a period of 3 to 4 year

FIG. 114. Ore Stock piles in Ely District. These Stock-piles contain the winter ac cumulation of ore to be shipped during the following open season (May to Novembe inclusive.)

according to whether the mine is running at small or at maximum capacity. Normal capacity for the mine is 720,000 tons a year. The block of ore between the 11th and 12th levels yielded roughly 2½ million tons.

There is practically no contract work at the mine. Drifts are run by machine Four men with two No. 3 Rand machines abreast operating 2 shifts make from 2(to 30 feet a week, averaging 24 feet in hard ore. Two muckers are on the job whe needed. Wages are $2.65 for machine men and $2.35 for helpers and muckers. A 9 by 10-foot drift requires from 10 to 12 holes; the lifters, three in number, being fire last. The track is 25 to 30-pound rails laid to a 24-inch gauge. Sub-drifts ar

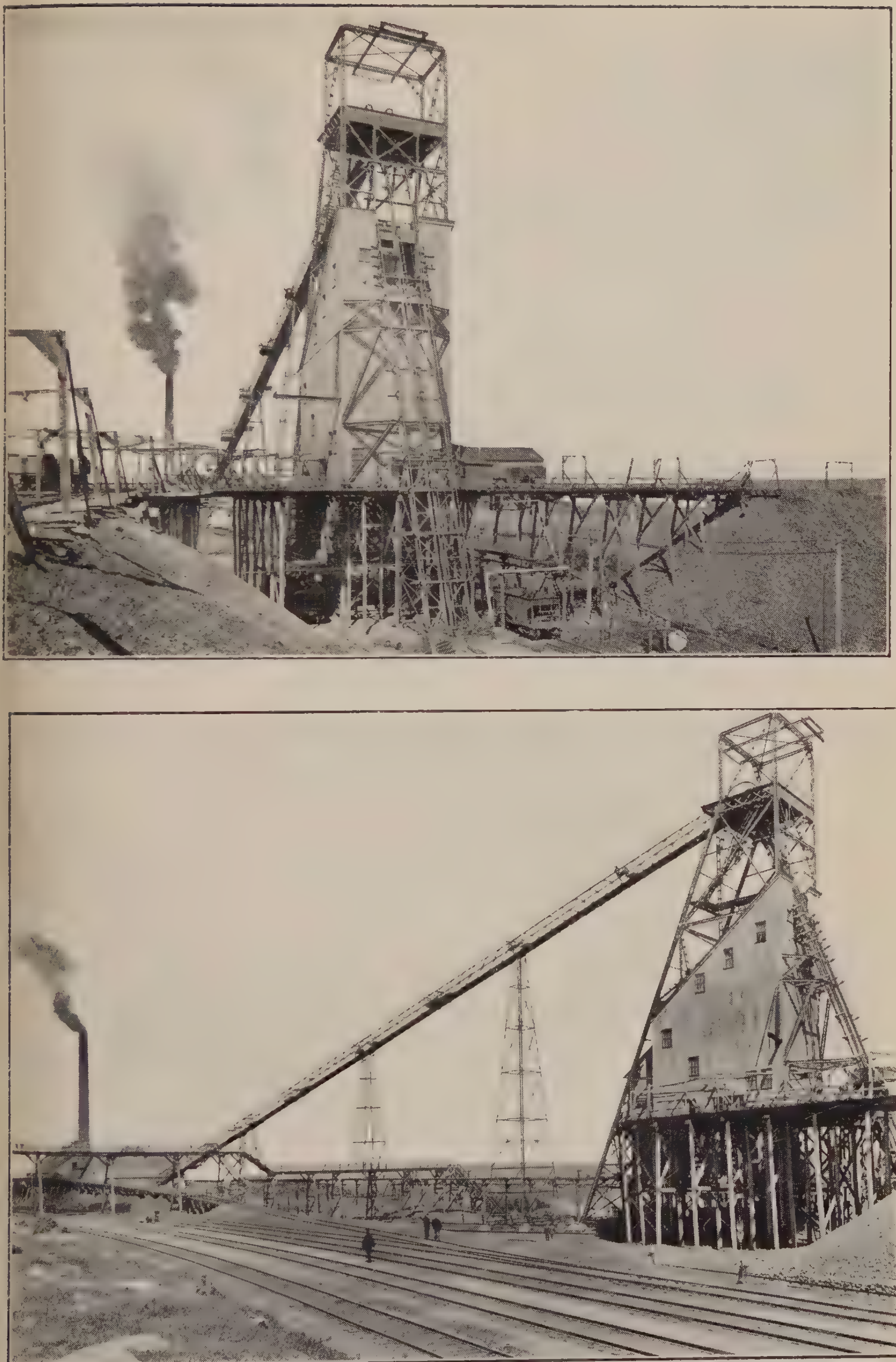


FIG. 115. Headframe Pioneer "A" Shaft.

FIG. 116. Old type of Headframe. Now largely supplanted by compact steel headframes as illustrated in Figs. 40 and 115.

similar to the main drift, though no track is laid. The distances are so short that it is cheaper to wheel.

Vertical raises are run 5 by 5-foot rock dimensions, cribbed 4 by 4 feet inside. They are run with a Waugh drill, 6 to 9 holes 1½ inches in diameter and 4½ feet deep. One miner ($2.65) can drill a round and put in 3 to 4 sets of cribbing in a shift. Inclined raises are run with a regular piston drill, two men, two shifts, rate of advance from 20 to 25 feet per shift, 5 by 6-foot rock dimensions.

The machine drill is rarely used in stoping or caving. In the crushed ground in the caves 1½ to 2-inch pop holes are rapidly drilled by hand, from 20 to 36 inches in depth, 40 per cent powder being used. A single miner is limited to 10 pops in one blast, using a stick of powder to a hole.

When blasting in subs immediately above the main level, the trammers in the level are protected by a man stationed near a "hello pipe." A white electric light in the sub, when turned on, indicates that he is at his post. The miner on the sub notifies him of proposed blast giving him the numbers of the chutes affected by the blast. The man on the main level turns off the light and hurries to the proper chute, remaining there until the blast is fired.

Tramming.—Seventy-five-pound rail, 24-inch gauge with 30-pound frog and switch rails are used. A 6½-ton Goodman motor hauls 12 to 15 cars in a train. The desired number of cars is uncoupled from the empty train as it returns. These cars are immediately filled by trammers and ready for the motor on its return from its next trip. At the shaft station the track passes over the ore-bin opening, there being a 2-foot dumping space on each side. The pocket man standing on one side and the brakeman on the other, armed with 4-foot bars, dump the cars without stoppage of train as it passes slowly over the pocket. The cars are regulation 3-ton gable bottom type with swinging side. Each side has a horizontal disengaging lever caught by a hook at each end.

The tramming crew consists of 25 men on the 12th level, day shift only, as follows:

2 motor men at $2.45
2 helpers at $2.35
16 chute pullers (loaders) at $2.45
4 track cleaners at $2.25
1 trammer boss at $2.75

On the 13th level development work requires 4 men tramming on each shift, 8 men all told:

1 motorman
1 helper
2 chute pullers

It is not possible to give the total operating force since the five properties in the district are under one organization. The office and engineering force, the machinists, blacksmiths, carpenters and their helpers serve the five mines. The force directly

chargeable to the two Pioneer shafts varies from 425 to 475 men and averages roughly 450 men. The labor classification sheet shows the following division:

11th Level—		
Miners working in caves	2 shifts	100
12th Level—		
Chute pullers (working in pairs)	1 shift	16
Motormen and helpers	1 shift	4
Track cleaners	1 shift	4
Timbermen	1 shift	4
Trammer boss		1
12th Level—		
1st sub, miners	2 shifts	60
2d sub	2 shifts	70
13th Level—		
1st sub—		
Timbermen	2 shifts	8
Machinemen	2 shifts	28
Muckers	2 shifts	12
Motormen	2 shifts	4
Track cleaners	2 shifts	4
Raising	2 shifts	6
Shiftbosses	2 shifts	2
14th Level—		
Machinemen	2 shifts	4
Pumpmen	2 shifts	2

GENERAL

A Shaft—		
Surface landers	2 shifts	4
Cage men	2 shifts	4
Skiptenders	2 shifts	4
Shaft repair men	2 shifts	4
Skip checkers in headframe	2 shifts	2
Trackmen	1 shift	2
Electricians	1 shift	2
B Shaft—		
Skiptenders	2 shifts	2
Skip checker	2 shifts	1
Men loading cars from headframe pocket	2 shifts	6

The tonnage per man on this work ranges from 4½ to 5½ tons and from 8 to 9 tons per man employed underground. During one month roughly 55,000 tons were produced with an average crew of 445 men working 26 days (no Sunday work), practically 4.8 tons per man.

FIG. 117. A "Cave" Lake Angeline Mine.

FIG. 118. A "Cave" Lake Angeline Mine.

FIG. 119. Re-inforcing Drift Timbering, Norrie Mine.

FIG. 120. Top of a Raise, Norrie Mine.

FIG. 121. Timber Crush, Norrie Mine.

FIG. 122. A "cave" in End of a Drift, Norrie Mine.

FIG. 123. "Scramming," Norrie Mine.

FIG. 124. General Underground View, Norrie Mine.

The preceding illustrations, Figs. 117 to 124 inclusive, are introduced to give a general idea of the caving method as it is applied on the older ranges of Michigan and Wisconsin. They are not to be taken as illustrating Minnesota practice.

THE SOUDAN MINE

The Soudan ore-bodies differ in many respects from those just described. The ore occurs in lenses from 150 to 1,000 feet long, 5 to 8 feet wide, and 200 to 500 feet along the vertical axis. There is a succession of these lenses one beneath the other, all dipping at from 65° to 75°. Mining is at present being carried on at a

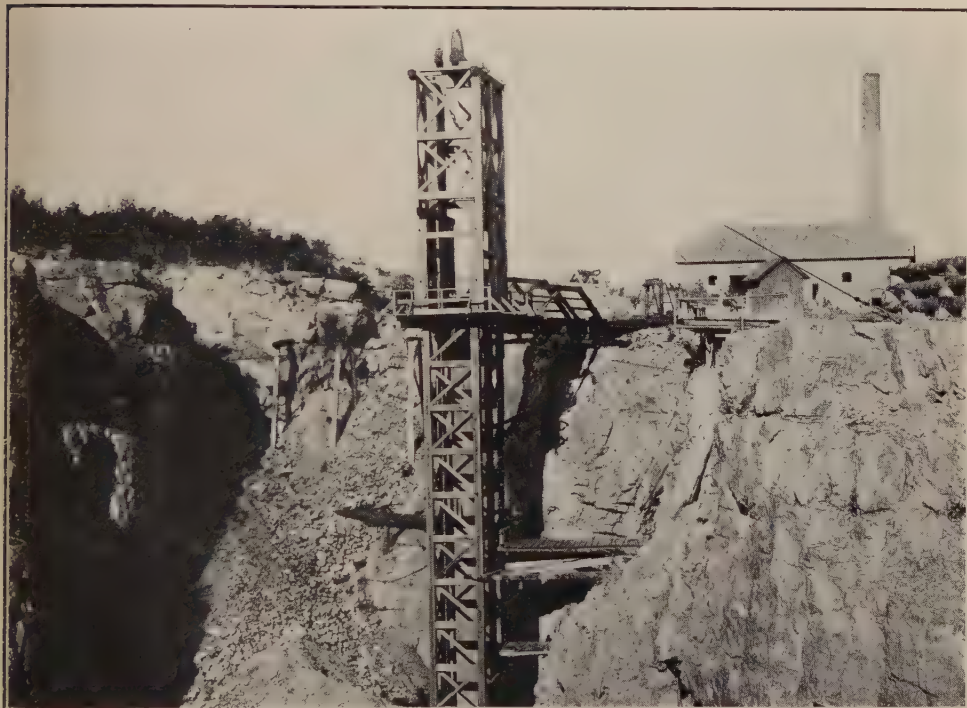


FIG. 127. No. 9 Shaft, Soudan Mine.

vertical depth of 1,250 feet. The most important difference between Ely and Soudan ores, from the mining standpoint, is the extreme hardness of the ore and the weakness of the enclosing walls of green schist (soap-rock).

Several deposits were started as open-pits and thus worked to a maximum depth of 150 feet, when the weak walls necessitated a change to underground methods. A combination of breast stopes 20 feet high, followed by underhand stopes of the same height, was tried and abandoned on account of the treacherous walls. A system of longitudinal "back-stoping" or overhand stoping with filling was adopted that proved satisfactory in every way.

The various steps in the development of this property are fully illustrated in Fig. 125, Plate X, which shows a cross-section at No. 8 hoisting shaft, a 3-compartment incline located in the footwall about 100 feet from the ore-body. Fig. 126, Plate X, is an idealized longitudinal and transverse section. The mine is opened by several shafts.

Levels are opened at 80-foot vertical intervals. A wide cross-cut is driven to and through the ore-body to the hanging wall. A breast stope is then started, full width of the ore-body with a 15 to 20-foot face. When this cut is sufficiently advanced to warrant the commencement of filling, 3-piece drift sets are erected and closely lagged

FIG. 128. Erecting Main-Level Sets Preparatory to Filling.

over; cribbed chutes 5 by 5 feet inside are run up at 30-foot intervals along the drift (see Fig. 128). Ore chute and ladderway are opposite each other. Broken rock is then run in and the opening filled to a height of 12 to 15 feet. It is desirable to have at least 5 feet of fill above the caps of the drift sets. A new slice is then started the full width of the stope, with a 10-foot face. The blast brings down large slabs that must be broken up before the ore can be thrown into the chutes. As the stope advances the chutes and manways are cribbed up and another layer of broken rock or filling is introduced. This rock is obtained by putting up raises in the footwall

close to the ore from the first level to the open-pit, from the second level to the first and so on. These raises are in the rock, one side being at contact between the wall and ore-body. As the back of ore is stoped out and the level of the fill rises, the side of raise at first formed by the ore is maintained by timber. The raises are cribbed through the different levels as work progresses. A raise may be tapped at any point in the stope by putting in a chute. This is illustrated in Fig. 129 which also shows the car used for distributing fill in the stope. The box can be swung in a horizontal plane so that the car may dump in any side or end position. Stoping may be going

Fig. 129. Drawing and Distributing Filling. Diamond drill at work in the breast on the left.

on at several levels and the same rock raises may be used for filling the stopes. Should rock be wanted in one of the upper stopes the raise may be filled to that point, a matte of timber inserted to prevent rock descending it, a chute put in, and rock may then be drawn as needed. Meanwhile the rock stored below this point may be withdrawn at the lower level if needed. The fill for the first few levels was obtained largely from the open pit by blasting down the sides. The floor-pillars of ore that must be left between the first stope and the old pit and between successive stopes are removed through footwall drifts run in at the same level as the top of the fill beneath

FIG. 130. Loading Car from Chute.

the pillar to be mined. From these drifts cross-cuts are driven into the ore-pillar on a level with the top of the fill. If the ore does not readily cave, it is made to cave and is then taken out with drift sets. The fact that the superincumbent rock freezes solid in the winter time simplifies the vein oval of the pillars. After the removal of the first level pillar the waste used to fill this stope may be drawn into lower stopes.

OPERATING DETAILS

Drifts.—Drifting and cross-cutting are generally slow and expensive work in this mine, as the ore is very hard and much of the country rock is hard jasper quartz. A 7 by 8-foot drift requires 12 to 14 holes. Two No. 3 Ingersolls working on each of two 10-hour shifts will advance these drifts from 20 to 30 feet a month. The material is so hard to drill that during the drilling the holes are often "bulldozed" or shaken up with a stick of powder every 12 to 18 inches. The work is done on contract and prices run up to $30 a foot in the jasper quartz.

Raises.—Raises are usually run in rock, 6 by 6 feet, on contract. Nine holes are usually put in with Ingersoll hammer drills. One man will drill from 25 to 30 lineal feet in a 10-hour shift. The monthly advance on 2 shifts is from 45 to 50 feet.

Stoping.—The ore lense is opened by a central drift about 7 by 8 feet. This is followed by a breast stope, 20 feet high, full width from foot to hanging, cut out by two or more machines according to the width of ore. Later a drift is timbered in, made of 9-foot posts, 11-foot caps, set 3-foot centers except at chutes where the sets are of 20 to 24-inch timber, lagged with 6 to 8-inch round lagging on top and 3 to 5-inch on the sides. Long sway braces are used to brace the sets. Cribbed manways (ladder and pipe) are built up on the hanging wall side at 50-foot intervals and ore chutes every 25 feet on the footwall side. Rock raises are run up from level to level in the footwall with one side at contact with the ore; their distance varies with the width of ore, at least 100 feet apart.

In one breast stope, 61 feet wide, 6 Ingersoll machines (2½-inch, 2 men to a machine) were drilling on two 10-hour shifts. The ore was very hard; the progress per machine varied from 6 to 10 feet per 10-hour shift. This 20 by 61-foot breast required on each of 2 shifts:

6 machine men ($2.65) and 6 helpers ($2.45).

6 trammers ($2.45).

1 man barring down and 1 ore shorter ($2.30).

The foreman on the level points all the holes. During one month this crew averaged 85 tons per 24 hours breast stoping. On succeeding slices, which are carried 15 feet high, the tonnage for the same crew is trebled; this illustrates the advantage of the second free face. In former years (as illustrated in Fig. 129) a Sullivan "E" diamond drill was used to drill long back holes in the stopes. In a 70-foot stope 3 holes would be drilled, 25 to 30 feet deep, the outer holes pointing slightly towards the hanging and foot on their respective sides. The drill makes about 12 feet headway in 10 hours. The hole is first sprung with dynamite and then charged with 50 pounds of 50 per cent dynamite. The blast would bring down from 500 to 1,000 tons of ore, while the cost per foot drilled was greater than with machine

drills; the progress was faster and the product per foot of hole greater, resulting in a lesser cost per ton. Within recent years the cost of diamonds has more than doubled and the diamond wear on this hard ore is quite heavy.

During an average month in 1910 this property produced 9,000 tons of ore for the month (52 shifts of 10 hours each), with a crew of 200 men in 24 hours:

10 machines stoping, 2 shifts.
7 machines drifting, 2 shifts.
1 hammer drill raising, 2 shifts.
6 timbermen, 1 shift.

The hardness of the ore may be judged from the fact that these 18 machines were using over 800 drills in 24 hours. Of the 200 men employed, 145 were underground; the tonnage per man underground per shift is 2.41; the tonnage per man per shift, including all men employed, is 1.54 tons. The Soudan Mine is at present shipping about 75,000 tons a year. Its maximum production was 592,244 tons in 1897. The total shipments aggregate practically 8½ million tons.

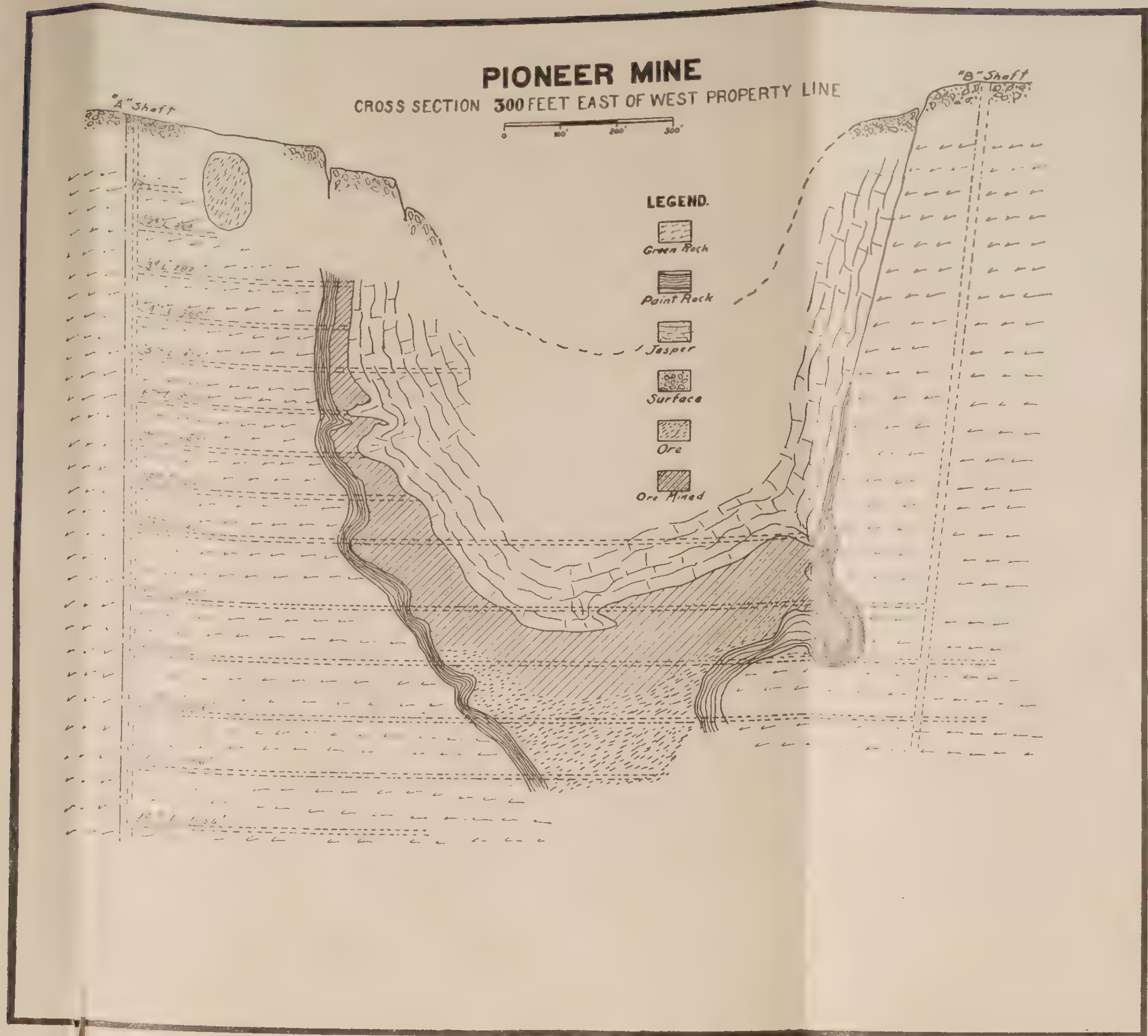


FIG. 104

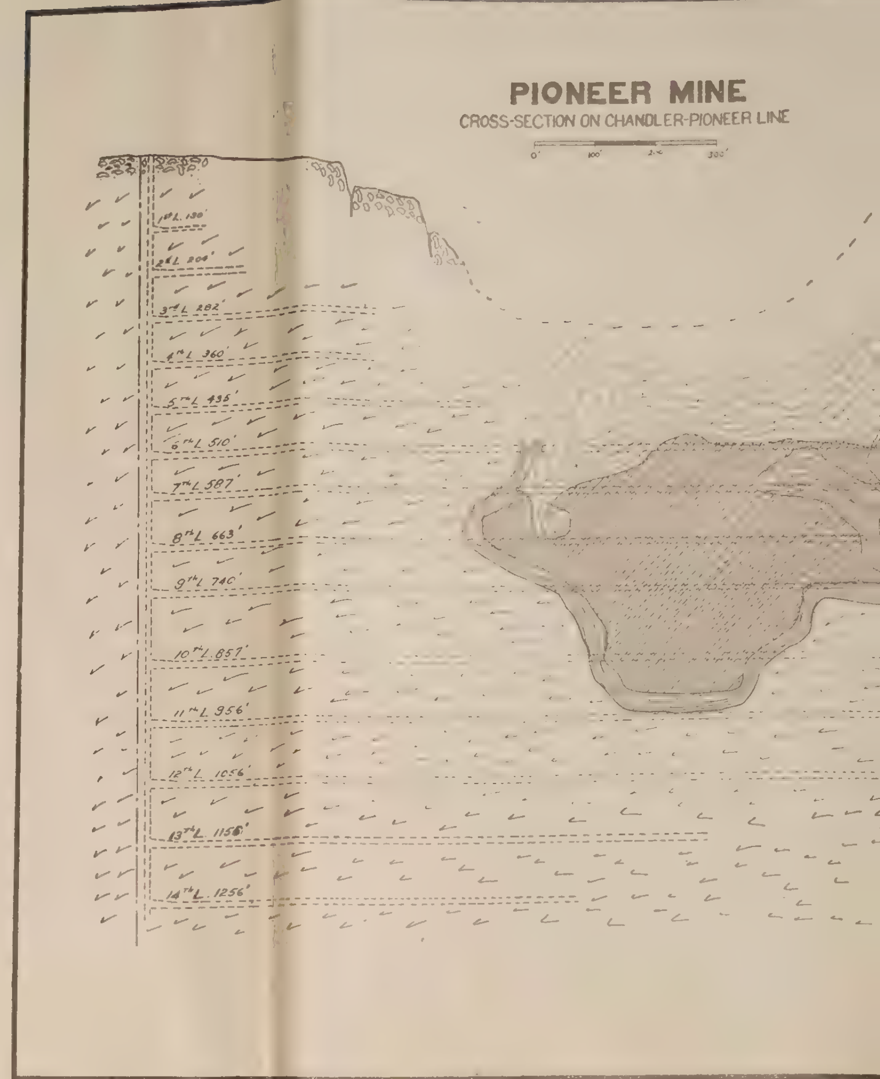


FIG. 105

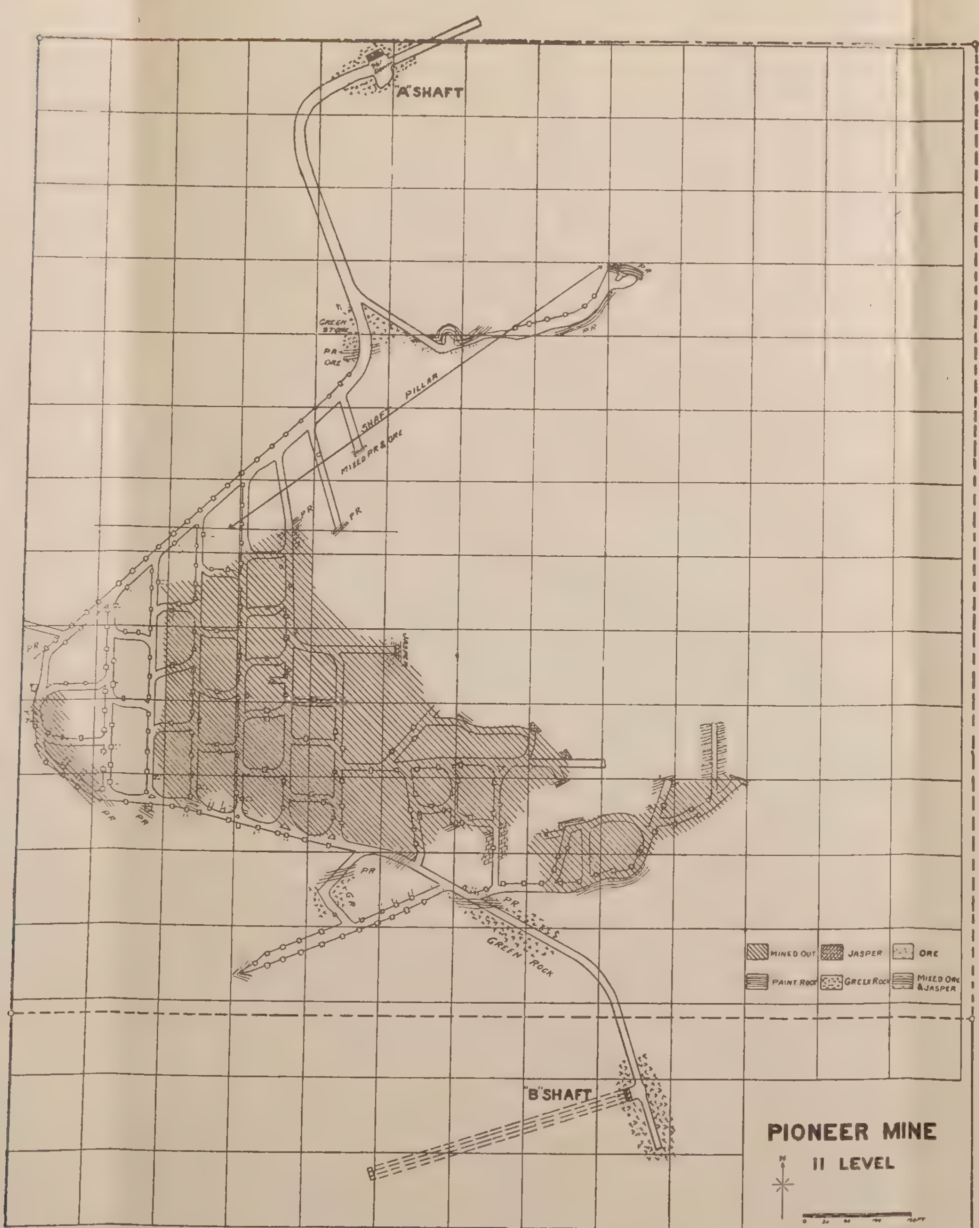


FIG 107—11th Level

FIG. 108—2nd Sub. 12th Level

FIG. 109—1st Sub. 12th Level

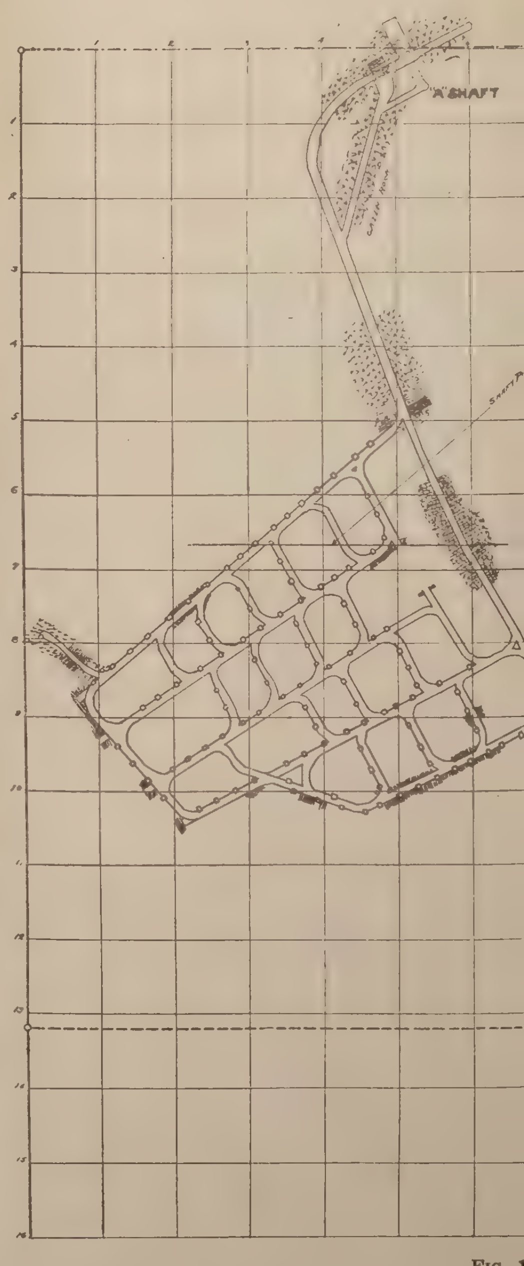
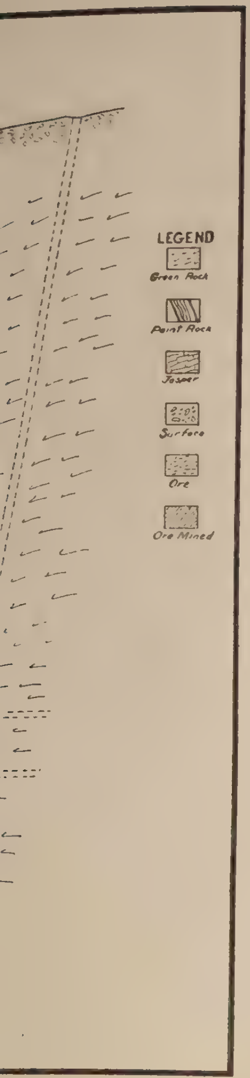


FIG. 1



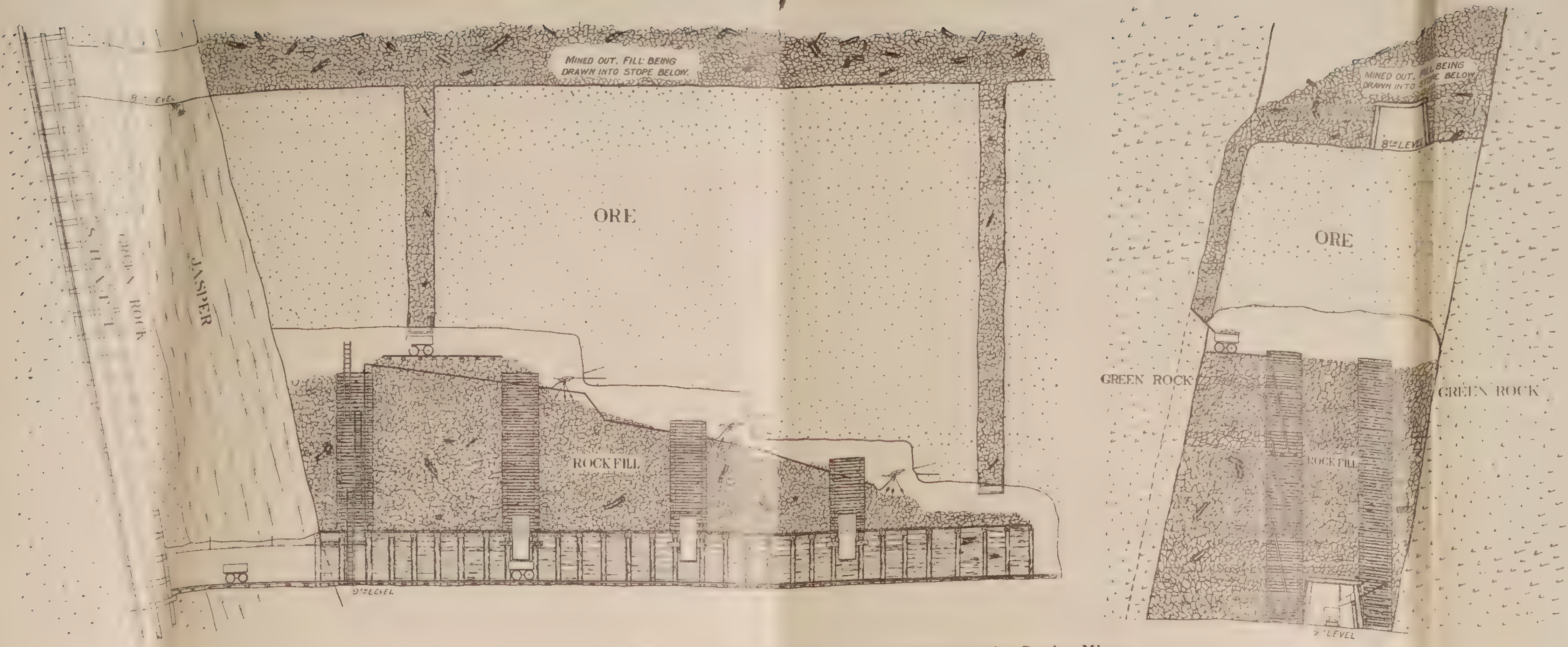


Fig. 126—Idealized Longitudinal and Transverse Section Soudan Mine

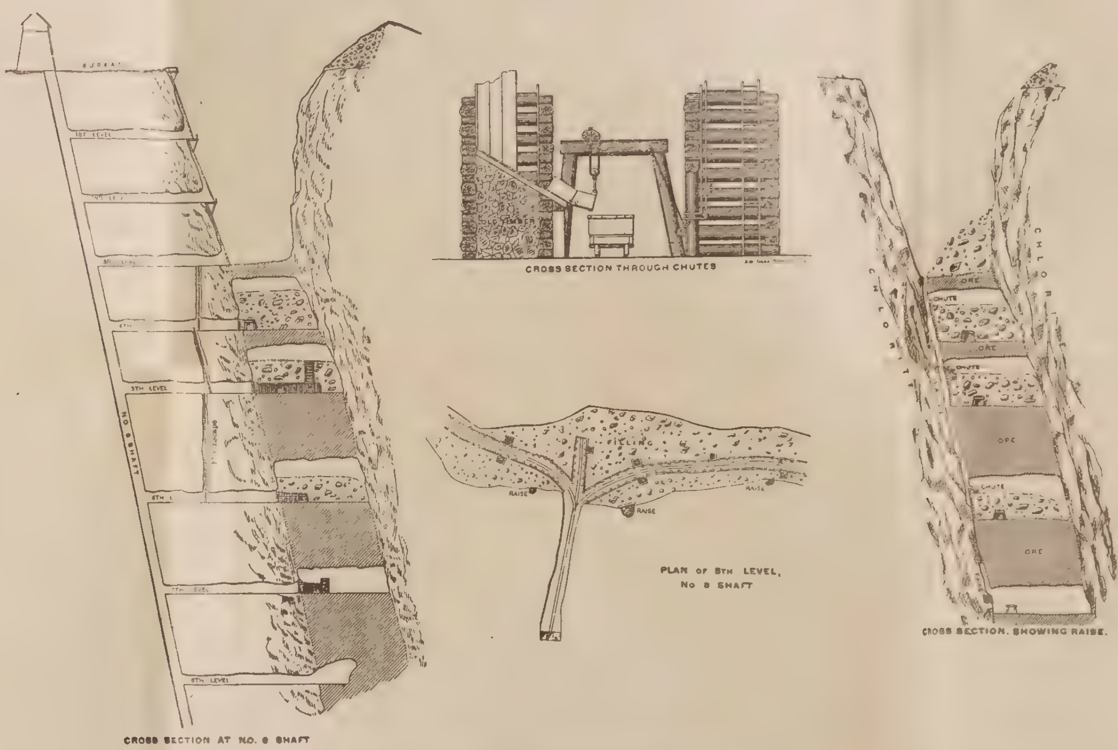


Fig. 125—Details of Soudan Mining Method

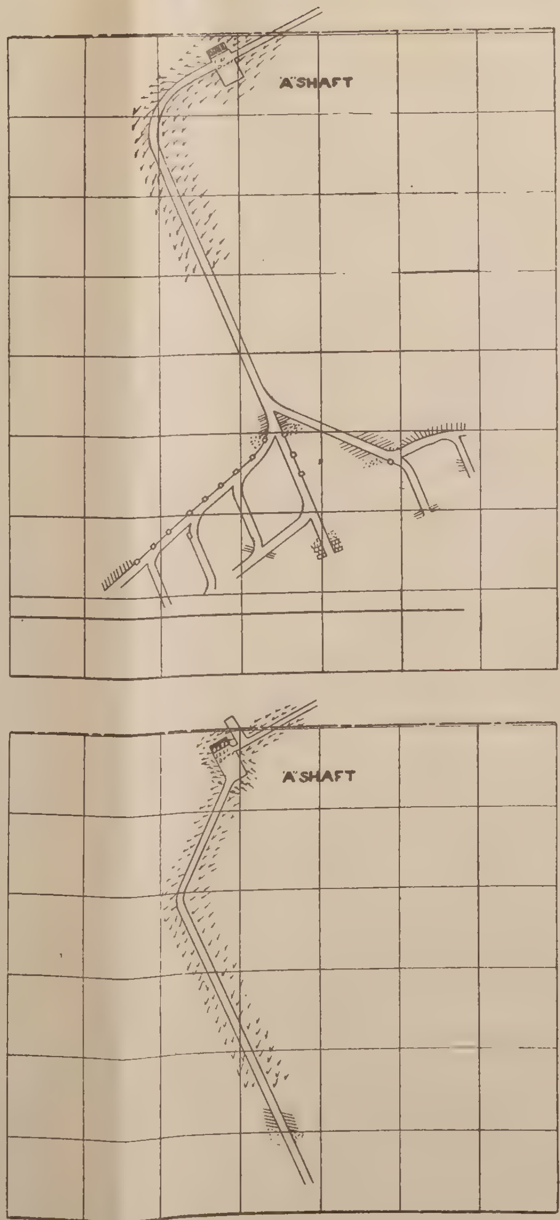


Fig. 111—13th and 14th Levels

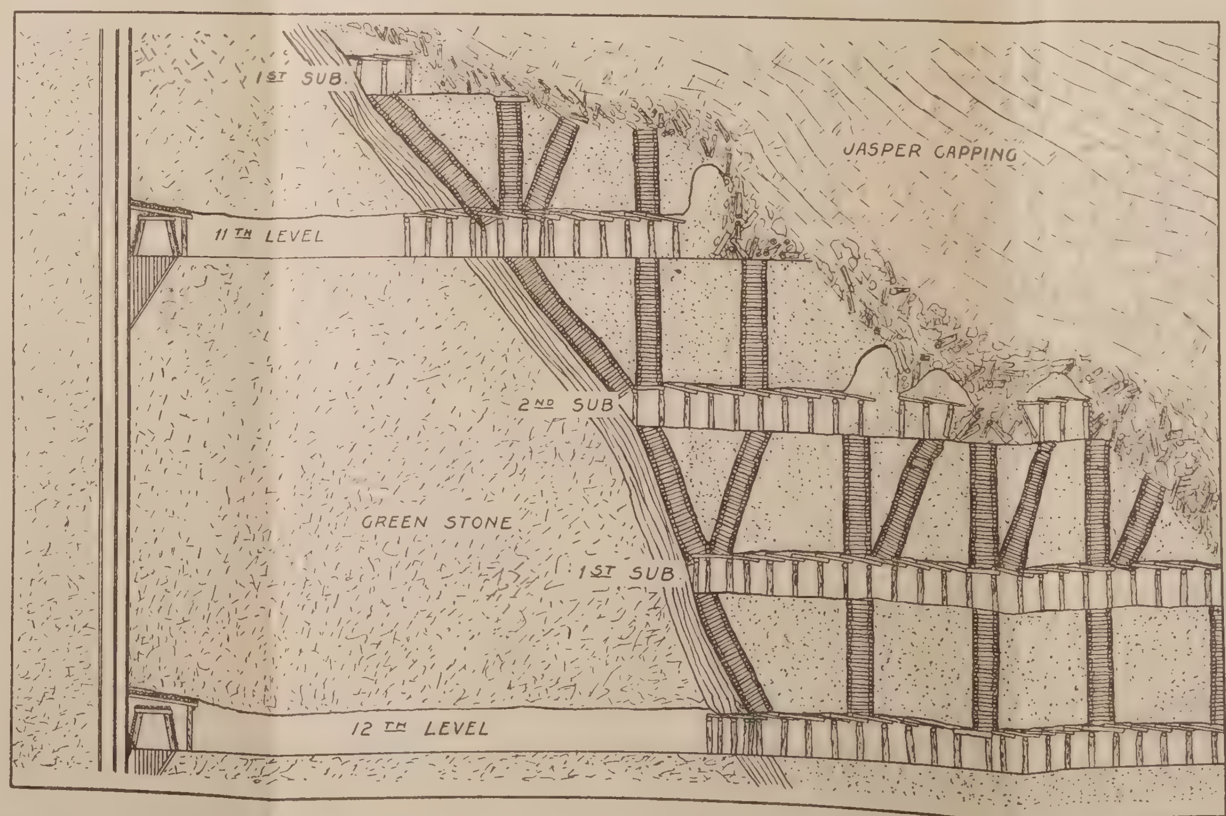


Fig. 106—Pioneer Caving System

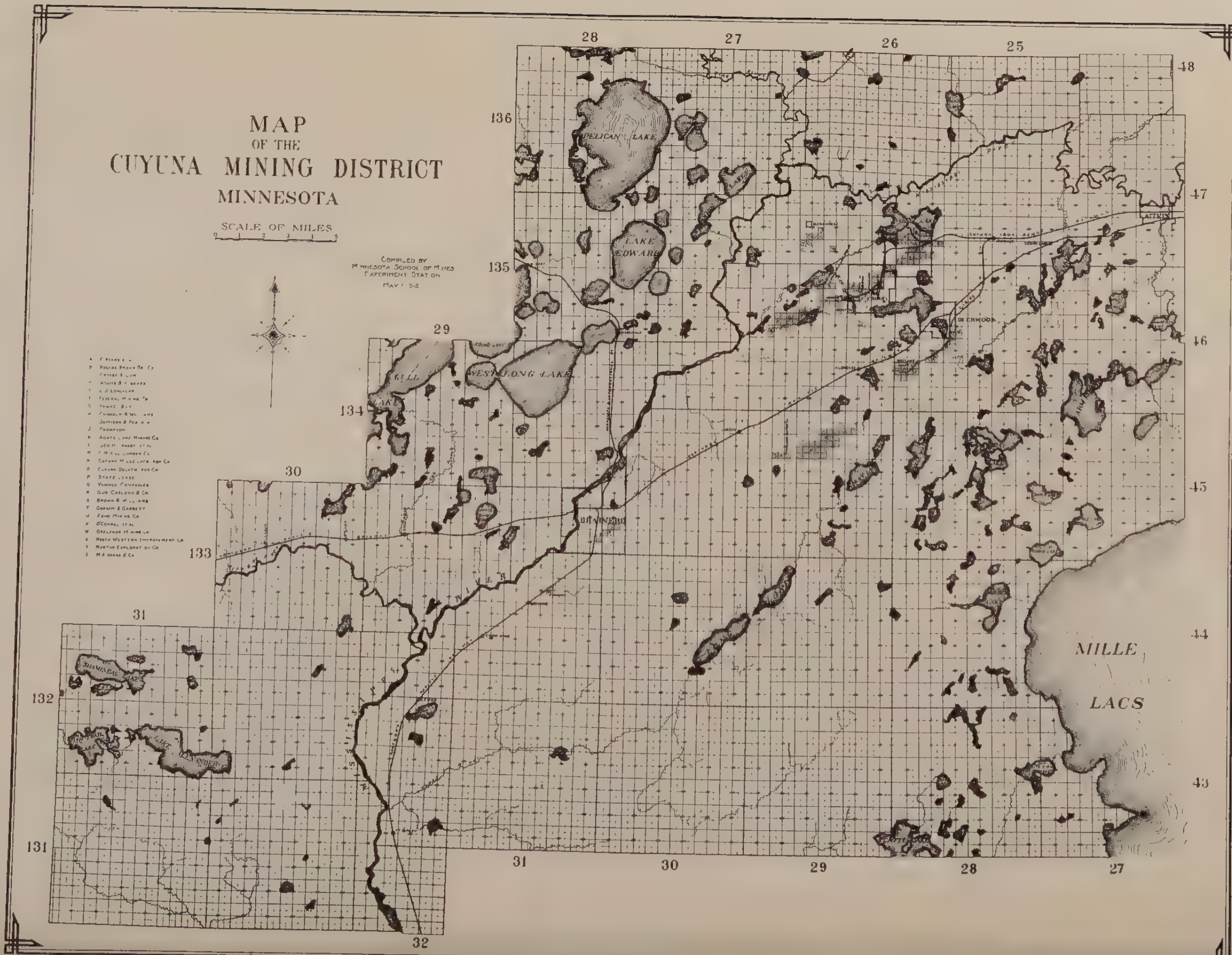
THE CUYUNA RANGE

The object of this publication is primarily to record current mining practice on Minnesota Iron Ranges, and the Cuyuna Range has hardly reached a stage to ...t of profitable discussion of its mining practice. The range is nearly seven years ... Considering the prospecting conditions, its development has been rapid. At present there are 5 producing mines and 3 more will be shipping shortly. The first shipment was made from the Kennedy Mine early in 1911. The shipments for the year ...gregate 147,439 tons. It is expected that shipments for the 1912 season will exceed ...lf a million tons.

Prospecting conditions were unusually difficult. There were no topographic ...atures to indicate the presence of the iron formation which is covered by a deep ...tt, devoid of mineral outcroppings. Magnetic surveying proved of material assist...ce in the early exploration work. It must not be thought from this statement ...at the ore-bodies on the Cuyuna are directly indicated by magnetic phenomena. ...he soft hematite ores of the Lake Superior district as a whole are non-magnetic; ...is is true also of the majority of the enclosing rocks. There are, however, cer...in magnetic slates closely associated with the Cuyuna iron formation, whose strike ...nd dips may be ascertained with sundial and dip needle. From the data thus ...tained the engineer may approximately determine the position of the synclines or ...roughs whose structure is favorable to ore deposition and direct his drilling operations accordingly.

The Cuyuna operators have many problems to face. Some of them are inde...terminate and must so remain until more drilling, and especially more lateral, under...ground development, shall give the necessary information. The ore-bodies, compared with the ordinary ore-body the Missabe miner is accustomed to, may be considered quite small. They are irregular lenses of moderate size, containing from a quarter of a million to 8 million tons. The ore is not continuous across the width of the lenses (which range from 30 to 300 feet). There are bands of good ore, lean ore and slate, contained within walls of schist and ferruginous slate. In some prop...erties larger ore-bodies of manganiferous iron ore occur, containing up to 40 per cent manganese. The market for this product is at present problematic.

Physical conditions limit the operators to underground mining methods. Shaft sinking is quite expensive on account of the depth of overburden, the heavy flow of water, and the frequency of heavy streaks of quicksand. Most of the shafts are concreted down to the ledge. The pumping problem during subsequent mining operations is as yet indeterminate. It is likely, however, to be quite an important item


MAP
OF THE
CUYUNA MINING DISTRICT
MINNESOTA
SCALE OF MILES
COMPILED BY
MINNESOTA SCHOOL OF MINES
EXPERIMENT STATION
PELICAN LAKE
LAKE EDWARD
WEST LONG LAKE
GULL LAKE
BRAINERD
DEERWOOD
AITKIN
MILLE LACS
SHAMINEAU
LAKE ALEXANDER
RIVER
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of expense at the average Cuyuna mine. It would seem probable that the "top-slice" method in general use on the Missabe will be best suited to the Cuyuna ore-bodies as a whole. In this connection the variation in grade and lack of continuity of pay ore in some of the ore-bodies will be an important factor. The Cuyuna ores have a very desirable structure from the smelting standpoint and are classed as "old range," with a preferential price of 20 cents per ton over the softer and finer Missabe ores.

Predictions as to the future of the Cuyuna Range have come from many sources, some are optimistic, others the reverse. The boundaries of the range are defined in a general way only, and the outlook for a decided extension of the ore-bearing area is favorable. The Missabe Range, with a lesser area than the Cuyuna, has something over 30,000 drill holes, while the Cuyuna has perhaps 2,500. These figures are merely estimates, but they will serve to indicate to some extent the vast amount of prospecting still to be done on the Cuyuna and the possibilities for adding to the 50 million tons of merchantable ore already developed.

TRANSPORTATION

The rapid development of the iron-ore industry in the Lake Superior district an especially on the Missabe Range would have been impossible without adequate an relatively cheap transportation.

Five railroads tap the iron ranges of Minnesota. The Duluth & Iron Range wa the pioneer in the ore-carrying trade in this state, being opened up in the year 1884 In 1892 the Duluth, Missabe & Northern carried the first ore from the Mountain Iro Mine. The Great Northern, by the control of the Duluth & Winnipeg and the Du luth, Mississippi River & Northern Railroad, entered the Missabe Range near Hib bing. All these roads transport ore from the Missabe Range. The Duluth & Iro Range handles all the ore from the Vermilion Range. The length of haul varie from 70 to 100 miles.

The table gives the general equipment of the ore-carrying roads necessary fo the handling of the enormous yearly tonnage:

Road	Mileage	No. engines	No. cars
Duluth & Iron Range..........	168	94	5,247
Duluth, Missabe & Northern....	284	92	7,178
Gt. Northern (Missabe Division)	250	46	4,300

The Cuyuna Range is tapped by the Northern Pacific and the Minneapolis, St. Paul and Sault Ste. Marie (Soo Line).

The Duluth, Missabe & Northern handles a large proportion of the ore from the Missabe Range. In addition to the fact that it has a very large tonnage to haul, its problem is complicated by the necessity for making and maintaining certain ore-grades. The railroad is expected to handle promptly the product from perhaps 150 shafts and steam shovels and to lead this into the holds of steamships holding from 5,000 to 13,000 tons, classified into not more than five grades, having a rigid analysis.

It is a rare occurrence to have a train carry ore of such analysis that it may be run straight through from the shipping point to the docks without being split up for mixing. This mixing is directed to a large extent from the Hibbing office of the Oliver Iron Mining Company. It does not admit of much flexibility for railroad and mining operations do not admit of delays; boats must be loaded promptly and each boat to be loaded must have a fixed tonnage for a certain draught; finally, each cargo must satisfy within a fraction of a per cent one of five guaranteed analyses.

The details of the operation are sufficiently interesting to warrant a fuller description. At the mine the loaded cars are hauled from the pit or from the shaft to the mine siding. A sample is taken representing 10 cars. When a train is made

and started toward its destination, the sample and corresponding car numbers are [s]t to the laboratory. The analyses are reported and entered on a "train sheet." [Th]e train conductor meanwhile has telegraphed back his train report, confirming the [car] numbers. The office now has the necessary data regarding this train made up of to 50 cars, sampled in groups of 10.

[T]he Hibbing office daily receives from the dock advice of the freight boats due [to] arrive in the next 24 hours, their capacity and the grade of ore they are coming [for]. The office now begins to lay out in the dock what are locally called "ore [blo]cks" and each block is given a number. The dock agent is expected to keep track [of t]he various blocks, he must know the pockets in which they lie and prevent blocks

FIG. 132. **Proctor Switching Yards.**

from being split up. The boat cargoes are then made up to order from these various blocks. The blocks are prepared so as to constitute a cargo of the desired average analysis for a boat of a given tonnage. A cargo may of course consist of one, two, or more blocks.

In addition to the main distinction made between Bessemer and non-Bessemer ores the five groups are made up according to iron, manganese, and silica contents. When the several groups of 10 cars constituting a train-load differ sufficiently in analysis to approximate different classifications, they are given different block numbers. Any ore half way between two classifications is given a block number. Whatever ore is left in the dock is re-numbered. Often an entire train-load is added to blocks

already being formed in the dock; the added ore, if both are off grade, is used to raise or lower the block analysis.

A cargo is rated as containing a certain number of units of each element, depending on the group and tonnage. The blocks are first approximated from estimated weights until the train arrives at Proctor, where all cars are accurately weighed. The blocks are then checked and corrected. When the Hibbing operator finally has his train blocked out he wires his orders to the Proctor Transfer; the blocks are switched out and new trains are made up and run through to the proper dock without any further switching. Fig. 132 shows the Proctor switching yards.

FIG. 133. Ore Dock Loading Pockets.

The number and size or tonnage of the blocks is dependent upon the number and tonnage of available boats. A block is apt to run off grade in taking care of the ore as it arrives. This must be carefully watched and checked before it becomes impossible to neutralize the block without adding more ore than any boat would hold. A number of "balance blocks" that analyze "on grade" must be kept in reserve to draw from in order to complete a cargo where the tonnage in the block proved insufficient. Obviously a boat must always take an entire block since these mixtures are to some extent mathematical rather than physical; that is, any part of a block might fall short of or exceed the requirements of the particular group which can be satisfied only by the average analysis of the entire block.

It may therefore easily happen that neither of two boats of different tonnage may be able to load although the dock may be stocked far in excess of their joint capacity. A third boat coming in several hours later might find just the right combination and therefore taken precedence over the other two. The opportunities for delays and mistakes are endless, yet the system works most satisfactorily.

Docks.—The docks are huge structures from 1,500 to 2,500 feet long, standing 60 to 80 feet above water. They are built on pile foundations that reach down 30 to 40 feet below water line. They must be strong enough to stand a traveling train-load of from 4 to 5 million pounds and to take up a pressure of from 2 to 3 million foot-pounds in the longitudinal bracing every time a train is stopped on the dock. The construction of the more recent docks is steel and concrete. The loading pockets are at 12-foot centers. The hinges of the chutes through which the ore is discharged from the pockets into the hold of the vessel are 40 feet or more above the water level (Fig. 132).

Some of the newer docks represent an investment of close to a million dollars. The Duluth, Missabe & Northern docks can handle 20 million tons during the shipping season. Following is a list of the docks at Two Harbors, Duluth, and Superior:

Duluth & Iron Range		Ore Docks at Two Harbors	
Dock	Length	Width	Storage
1	1,388 feet	49 feet	40,400 tons
2	1,280 feet	49 feet	41,600 tons
3	1,054 feet	49 feet	34,000 tons
4	1,042 feet	49 feet	36,960 tons
5	1,050 feet	49 feet	35,450 tons
6 (Steel)	920 feet	53 feet	43,246 tons
			Total, 231,656 tons

Duluth Missabe & Northern		Ore Docks at Duluth	
2	2,336 feet	49 feet	69,120 tons
3	2,336 feet	59 feet	80,640 tons
4	2,336 feet	57 feet	119,274 tons
			Total, 269,034 tons

Great Northern		Ore Docks at Superior, Wis.	
Dock	Length	Width	Storage
1	2,244 feet	62 feet 8 inches	100,980 tons
2	2,100 feet	62 feet 8 inches	94,500 tons
3	1,956 feet	62 feet 8 inches	88,020 tons
			Total, 283,500 tons

During the busiest part of the shipping season trains arrive on the docks on half-hour schedules and it requires assurance and skill to empty a train-load into the right pockets and get out of the way for the next train.

Vessels are loaded entirely by gravity and a 10,000-ton modern freighter can be loaded in a couple of hours actual loading time. It usually takes 6 to 8 hours to load an 8,000 to 10,000-ton boat. The development of rapid unloading machinery has done more than anything else towards making possible the present stupendous movement of freight on the lakes. Up to 1880, vessels were unloaded by small bucket hoists and wheelbarrows. In 1880, A. E. Brown developed a single elevating and conveying cable. In 1882, an improved plant, consisting of five of these rigs in one

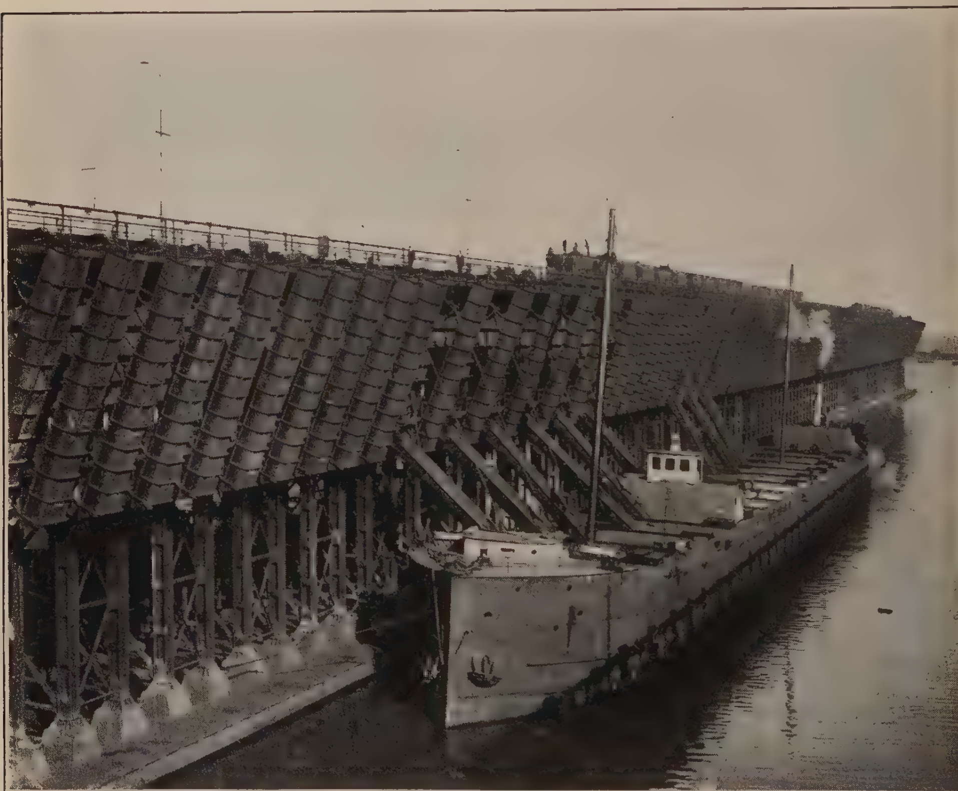

Fig. 134. Duluth & Iron Range Docks at Two Harbors.

house, was installed at Cleveland. The front pier of these machines was moveable and they were the first moveable pier cableways built in this country. Since then this and other devices have been greatly improved and the actual cost of unloading is about one-tenth of what it was by the bucket-hoist and wheelbarrow method.

Boats.—The improvement in dock appliances naturally led to similar improvements in ore-carrying boats. The first advance of note is the construction, in 1900, by A. B. Wolvin, of four 500-foot steamers. This was followed, in 1904, by a 10,000-

ton steamer, 560 feet long, 56-foot beam and 32 feet deep, containing 33 hatches, spaced 12-foot centers, built entirely on the girder system. This construction eliminates main deck beams and stanchions, thus leaving the hold unobstructed and giving the unloading machine free play upon the cargo. A cargo of 9,945 tons was discharged from this boat in 30 minutes, employing 8 unloading machines. The ore-carrying fleet numbers 550 vessels. The latest steamers are 600 feet in length and carry up to 13,000 tons; they represent an investment of half a million dollars.

Great care must be exercised both in loading and unloading these large boats in order that there may be no straining during either process. The dockman's concern is to load or unload the vessel promptly regardless of strains. The mate of the vessel

FIG. 135. Unloading Machinery.

has direct charge of the work and the care of the ship devolves entirely upon him. Fig. 133 shows a boat loading at the Duluth & Iron Range docks, and Fig. 134 shows the unloading machinery.

COST OF TRANSPORTATION

The cost of lake transportation from upper to lower lake ports has recently dropped from 60 to 50 cents per ton. This includes lake transportation and unloading at lower lake terminal. The cost of this latter operation dropped from 20 cents to 15 cents in 1910; and in 1912, by order of the Interstate Commerce Commission, to 8 cents per ton.

The cost of transportation from the mines to upper lake ports, including rail charges, storage and unloading into freight boats, was recently reduced. The following table gives the old rates and the rates now in force, while the third column gives the total transportation charge from the mine to the lower lake ports:

District	Rail and dockage to upper lake ports		Total transportation charge from mine to lower lake ports	
	1892 - 1911	1912	1911	1912
Ely	1.00	.60	1.60	1.10
Tower	.90	.60	1.50	1.10
Missabe	.80	.60	1.40	1.10
Cuyuna**		.65		1.15

**The Cuyuna rate will be decreased to 60 cents when the season shipments shall have attained 2 million tons.

IRON-ORE PRICES

The price of lake iron ore is always quoted f. o. b. lower lake ports and all contracts are so based. The Lake Erie price is based on the "Valley Price," that is, the price paid at the furnaces in the smelting districts around Pittsburg and in the Shenango and Mahoning Valleys. The Lake Erie prices are uniformly 60 cents under the Valley prices. The latter are subject to decided fluctuations. In 1898, Missabe non-Bessemer was worth, at Lake Erie ports, \$1.75, and Missabe Bessemer, \$2.25, with a \$1.45 transportation charge. In 1907 the prices were respectively \$4.75 and \$4.00, with \$1.56 transportation charge. The accompanying table gives the 1911 and 1912 Lake Erie prices for standard Bessemer and non-Bessemer ores with their guaranteed analyses. The "Base Unit Value" is based on "Valley Prices" as given in the table. Therefore, the f. o. b. Lake Erie price for any ore may be found by multiplying the Base Unit price, given under its proper classification, by the percentage of natural iron and deducting 60 cents (rail charge to Valley smelters):

Standard grade	Natural iron	Phos. %	Moist. %	Dried Iron %	Year	Lake Erie	Valley	Base unit value
Old Range—Vermilion..					1911	4.50	5.10	0.0927273
Bessemer	55.00	.045	10	61.12	1912	3.75	4 35	0.0790909
Missabe					1911	4.25	4.85	0.0881818
Bessemer	55.00	.045	10	61.12	1912	3.50	4.10	0.0795454
Old Range—Vermilion..					1911	3.70	4.30	0.0834951
Non-Bessemer	51.50		12	58.52	1912	3.00	3.60	0.0699029
Missabe					1911	3.50	4.10	0.0796116
Non-Bessemer	51.50		12	58.52	1912	2.85	3.45	0.0669902

In Hurd's Iron Ore Manual, published by Rukard Hurd, Secretary of the Minnesota Tax Commission, will be found a very complete discussion of this question in all its phases.